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THE

## CYANIDE PROCESS

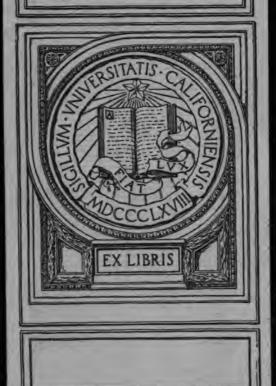
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# EXTRACTION OF GOLD

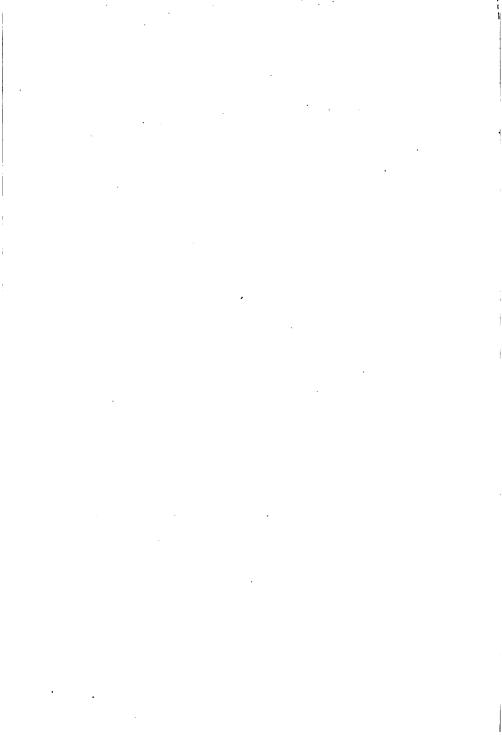


M. EISSLER

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THE

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THE

## CYANIDE PROCESS

FOR THE

## EXTRACTION OF GOLD

AND

ITS PRACTICAL APPLICATION ON THE
WITWATERSRAND GOLD FIELDS
IN SOUTH AFRICA

BY

### M. EISSLER

MINING ENGINEER; A.I.M.E.; MEMBER OF THE INSTITUTE OF MINING AND METALLURGY AUTHOR OF "THE METALLURGY OF GOLD." ETC., ETC.



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1895
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TATIO

### INTRODUCTION.

It is a great satisfaction to me to be in a position to place before the public, in the present volume, an account of the Extraction of Gold by the Cyanide Process, which I believe will be found by Metallurgists who desire to avail themselves of the process to be sufficiently full and complete for practical purposes. That I am in a position to do this, is owing to the fact of my having made a lengthened stay in the Witwatersrand gold fields — whence I have just returned—and to my having enjoyed there unique opportunities of studying the process in actual operation.

Before entering upon an account of the process, some further introductory remarks will be advisable.

Metallurgy, as applied to preparing and winning the gold from its ores, resolves itself, broadly speaking, into two methods, in one of which mechanical

means, and in the other chemical means, are employed. I consider the stamping or crushing of the ore in a battery, and the catching of the gold on amalgamated copper plates, a mechanical process, although the recovery of the gold in chemical combination with mercury might entitle it to be considered otherwise. The whole process is nothing more or less than a simple system of concentration, as the gold, owing to its great specific gravity, separates from the lighter gangue in which it is enclosed; and when the pulverized ore particles are washed away, the gold grains sink and are arrested on the amalgamated smooth surfaces of the plates over which the stream carries them, or get entangled in the hairs of the blankets, or are stopped in the riffles of the sluice box, or whatever other method is used in the battery to save the free gold.

When gold is in combination with other minerals, more complicated methods have to be employed to collect it by mechanical means, but these it will be unnecessary to describe in the following pages.

There are very few chemical means by which gold can be won from its ores, leaving aside the smelting process by which the gold is collected in the smelting furnace by means of, and in combination with, other metals. The reason why gold is not easily won by chemical means is because it is a non-oxidizable metal, which maintains its purity, and is therefore mostly found in its native or metallic state. Owing to its rarity it has become of such great value, and the medium of interchange since the remotest times.

Its solvents are aqua regia, chlorine, and potassium cyanide. The application of such a corrosive agent as aqua regia for the treatment of ores on a large scale is out of the question, and of the chemical means at our command there remain only the last two. By exposing the gold ores, or the concentrated portion of the same, to the action of chlorine gas, the gold is converted into a soluble chloride of gold by the well-known chlorination methods; and by the other method the gold is dissolved in solutions of cyanide of potassium.

That gold when in a fine state of division was soluble in cyanide of potassium, was already known in the middle ages, and the gilding of metals was carried out in those remote days by jewellers and alchemists, by the use of gold in cyanide solutions. Of course, gilding by means of fire was usually employed.

Several scientific books, dating back to the beginning of this century, mention the solubility of gold in potassium cyanide solutions. The application of this solvent for the treatment of auriferous ores was first thought of and patented in the United States in 1867; and although the process was tried and experimented with by some eminent metallurgists of that country, no practical or commercial results were obtained.

Now, how is it—in spite of repeated failure in America and elsewhere—that within the last few years the cyanide process has come into such preeminence, and as such a grand success, before the metallurgical world?

In answer to this question, I believe I am safe in stating that the ores of the Witwatersrand gold fields, where the cyanide process was first introduced, carry the gold in a pure and metallic state, in an extremely fine state of division (and even in the pyrites, the gold does not occur in chemical combination, but in a free state); and, therefore, all the conditions existed there to make the application of the cyanide process a perfect success. These facts are not only of interest to the metallurgist, but they should also throw some light on the geological features and conditions under which these peculiar conglomerate beds were formed.

The mines of this district, it should be said, will for ever have to acknowledge the immense services which the MacArthur-Forrest Company have rendered by developing and introducing the process, as they spent large sums to bring it into practical shape, and to demonstrate its commercial advantages. But for those exertions, enormous values would have remained practically unavailable, preventing many mines from working at a profit, and the production of these fields would not be what it is to-day by 50 per cent.

In this connection I can mention that out of a total gold production of 1,478,470 oz. in 1893, there were produced from the tailings 330,510 oz. by the cyanide process; and in August, 1894, out of the monthly production of 174,977 oz., nearly 58,000 oz. were won by the cyanide process.

The great future and potentiality of development which may be anticipated for these gold fields is strikingly illustrated by the opinion of such an eminent authority as Mr. Hamilton Smith, who (in his report to the house of N. M. Rothschild & Sons), has stated that that portion of the Witwatersrand which lies between the Langlaagte Block B and the Glencairn Mine—or about 11½ miles in length, along the strike of the main reef series,—if exploited to a verti-

cal depth of 3,000 ft., would yield in gold 215 million pounds sterling; while Bergrath Schmeisser (in his report made to the German Government), taking for estimate a depth of 800 metres, arrives at the yield of 208 million pounds, and for a depth of 1,200 metres, the yield of 349 millions. And when we consider that at least one-third of these prodigious amounts will be won by the cyanide process, one can hardly overestimate the importance of the work which the MacArthur-Forrest Company has done in bringing that process into its present position of prominence in the Witwatersrand gold fields. Certainly it cannot be gainsaid that they are entitled to a fair reward for their labours.

When I arrived in 1890 on the Witwatersrand gold fields, I undertook numerous experiments for the treatment of tailings by pan amalgamation, but with unsatisfactory results. The difficulty I had to contend with was the formation of large quantities of iron amalgam in the pan. The quicksilver became dirty and floured, causing very heavy losses, and I had a costly process before me to separate the iron from the gold; and although others followed me, and put up various devices for treating the tailings, it was only the advent of the cyanide process which solved this

most difficult problem, as pan amalgamation would have been too expensive.

So far, the process has not achieved similar success in other countries, which to my mind proves that the ores on these fields contain the gold in a free condition; and to further corroborate this view, I can state that at the Simmer and Jack mine the pyritic concentrates which are daily collected on the blankets yield by pan amalgamation 65 per cent. Of course these blanketings contain a considerable amount of free gold which escapes from the copper plates.

Within the last few months, Messrs. Siemens and Halske have successfully introduced on the same fields their patented process, which consists in precipitating the gold by electricity on sheets of lead; and, owing to certain economical advantages, it may be anticipated that their process will prove a formidable rival to its predecessor. In the succeeding chapters will be found an account of the working details of these processes, including some of the observations and researches of those gentlemen who have been most prominent in bringing them to their present state of perfection.

I must not conclude these remarks without expressing my deep obligations to the several gentlemen—

whose names will be found duly recorded in the following pages—to whom I am indebted for much of the material and information embodied in the volume, and for the opportunities so freely afforded me, during my stay in the Witwatersrand gold fields, of making myself practically acquainted with the working of the cyanide process. It is not myself only, but all who are interested—whether as metallurgists or investors—in the gold-mining industry, who have thus been laid under obligation; and as I have taken no small pains to put to good use the material and opportunities which have been so freely placed at my disposal, I trust that the outcome of my efforts in the present volume will prove thoroughly acceptable to that numerous body.

37, BELSIZE PARK,
SOUTH HAMPSTRAD, LONDON, N.W.
December, 1894.

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### THE

## CYANIDE PROCESS

FOR THE

### EXTRACTION OF GOLD.

### CHAPTER I.

#### ERECTION OF A CYANIDE PLANT.

Planning the Works.—In the erection of a cyanide plant, before planning the same, some essential points have to be considered. Are the works to be erected to treat an old accumulated stock of tailings? Or have they to be laid out to treat the tailings as they come from the battery? In many cases both these points have to be combined.

As tailings reservoirs are generally situated on the lowest point below the battery site, no provision exists below such dams for the erection of works which would permit of further dumping-ground and handling of the stuff by gravitation. At all events, the topographical conditions of Witwatersrand would allow such an arrangement only in rare instances. In localities where the fall of the ground below the reservoirs permits of the erection of the works, I would recommend this to be done, as it permits of the charging of tanks and their discharging by gravitation. In most cases the opposite course had to be resorted to on these fields; the tailings from the old pits or reservoirs had to be hauled up-hill to the cyanide works, steam power being mostly used. The arrangement is simple enough, as the dumping cars are pulled up on an inclined trestle-work above

the leaching tanks, and after discharging their contents they run back by gravitation, and are held back by the brake of the hauling drum. In large works five to six trucks, holding 20 cubic feet each, are hauled up at a time. At every mine the mechanical arrangement for the filling of the tanks is different, depending on local conditions. Messrs. Fraser and Chalmers have lately introduced a system of mechanical haulage by means of endless wire ropes which works very well, and which I would recommend in preference to anything I have seen on these fields.

To work old tailings by the cyanide process offers no difficulty to percolation, as they come to the works in the proper condition. They were cleaned of the slimes by the natural system of concentration, which takes place in the collecting reservoirs. It is very interesting to stand at the discharge end of the launder carrying the tailings to the reservoir, and to see how the tailings arrange themselves according to the natural laws of gravitation, and are prepared here for subsequent treatment. At the head of the tailing pit the coarser tailings accumulate, and near the dam the finest, and also slimes. The overflow from the first reservoir is collected in a second reservoir, where the slimy, clayey residue accumulates, which, strange to say, is as rich, and even richer, than the tailings in the first reservoir.

To lay out plans for an accumulated stock of tailings offers no great difficulty, provided there is near by a sloping ground permitting of the discharging from the leaching tanks and their dumping by gravitation. If the country is flat, the re-worked tailings will have to be hauled up an incline again and then dumped. On a flat site it will be necessary to place the leaching vats on masonry sufficiently high to give room for discharge, and gradient for the flow of the leaching solutions to the precipitation boxes.

When works are erected to treat tailings which are discharged from the battery, important appliances have to be resorted to, to prepare them for the cyanide treatment, and before they are collected in the leaching vats, owing to the physical condition of the powdered ore.

The discharge launder which carries the tailings from the battery to the cyanide works should have a grade of at least 3 ft. 6 in. in the 100 ft. to insure a good flow. In a flat country where no grade exists, the tailings should be elevated by means of bucket wheels to the proper height. From the experience gained on these fields, tailing pumps have not given satisfaction; it may be that they were not properly constructed, as I am told that in Australia they are in various places in successful operation. There are on several mines here large tailing wheels in use, and I should consider them the best way of elevating tailings, as they require very little attention and repairs when properly constructed and set.

Supposing that we desire to erect a cyanide plant directly behind a battery, the following grade would be required for doing the whole work by gravitation. Supposing the plant to be located 100 ft. from the battery—

							Feet.	Inches.
The grade for the dis	charge	launder	s will	l requ	ire	•	3	6
Masonry for settling	tanks.	•					6	6
Settling tanks .							10	o
Masonry for leaching	tanks						6	6
Settling tanks .							10	0
Precipitating boxes a	nd grad	le for o	utflov	v pipe	s.		6	O
Total grade							42	6

To this could be added from 6 to 10 ft. of grade for the storage tanks holding the cyanide solutions, wash and alkaline waters, but these are differently placed, and a lack of further grade would present no difficulties, as will be explained later on.

Fig I (Plate I.), showing a section of the cyanide plant of the Princess works, illustrates such a mode of arrangement as is here described.

Slimes.—The conglomerates on these fields, after stamping, contain a very large per cent. of slimes. Under slimes are understood the very fine particles of talcose and clayey material mixed with the very fine grains of quartz, iron oxides, and sulphides. If the whole of this fine material be allowed to collect

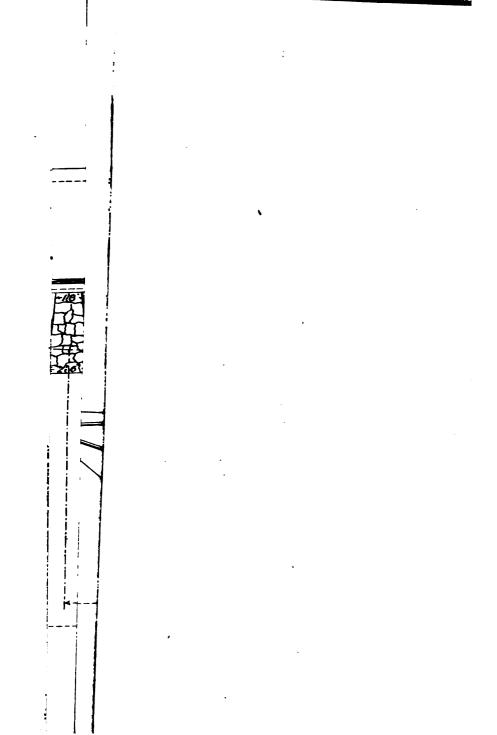
with the coarser grains, the percolation of fluids through the mass becomes impossible, and, therefore, mechanical means have to be adopted, aiming at a separation of the slimes from the coarser material.

Two methods have been introduced on these fields, aiming at the elimination of the slimes. The one by direct filling is the system introduced by Mr. Henning Jennings, the well known mining engineer; and the other the intermediate filling adopted by Mr. Charles Butters and Captain Mein, the manager of the Robinson mine.

I will take occasion to remark here that the appliances for the plant were materially changed by Mr. Charles Butters, who has done a great deal for the advancement of the metallurgical treatment of the ores on these fields, as he has introduced a great many practical details, all tending to lessen the cost of the process, and I consider it a very pleasant duty on my part to express my appreciation of his labours and of the good he has done by his work.

The elimination of the slimes has an economic bearing on the gold mining industry of these fields, when it is considered that at least 30 per cent. of the Witwatersrand ores, after crushing, pass away into slime pits: therefore, at the present production of 250,000 tons of ore per month, 75,000 tons go into the slime pits. At the present rate of progress it is almost certain that the tonnage will increase to nearly double this amount, and that within three years the Witwatersrand will be producing 300,000 ounces of gold or one million sterling monthly. If we take the average value of the slimes all around at only 5 dwts. per ton, this represents nearly 20,000 ounces of gold which goes into the pits monthly.

Up to the present no cheap method has been devised to deal with the slimes, so as to win the gold from them at a profit. The question of treating the slimes successfully is simply a mechanical one, as there is no chemical difficulty in the way: on the contrary, the solution of the gold can be easily effected, owing to the fine state of division in which the gold exists in the slimes.





My impression is that the eventual solution of the problem will be a system of agitation in vats fitted with mechanical stirrers, and after agitation and settling the solution will be decanted and a weaker solution added again, again stirred and decanted, till the last wash waters will only show traces of gold. This method will become applicable owing to the introduction of the Siemens and Halske process, which effects the precipitation of the gold from the extreme weak solutions which will have to be used. A successful solution of this problem will add largely to the gold production of the fields.

The Effect of Stamping on the Ore.—When ores are stamped in a battery, the resulting product is very uneven, and this is one of the recognised disadvantages of the batteries in use in all gold countries.

The majority of the mines on these fields crush fine, using mostly screens of 900 mesh per square inch, and about 50 per cent. of the ore is converted into slimes. To illustrate the fineness to which stamping reduces the ore I may mention that Mr. Williams, the metallurgist of the Crown Reef mine, informed me that the slimes which flow into their pits do not settle in 24 hours, and that if the overflow is taken and allowed to settle, these second settlings will assay 6 dwts, per ton, being richer, therefore, than the first settlings. He further stated that after the water leaves the second slime pit it still holds in suspension 2 per cent. of the total gold contained in the ore as it comes from the mine. This is the result from careful experi-In regard to the difficulty of settling the ments and trials. fine slimes which remain in suspension, it was ascertained at the Paarl Central works, that the overflow from a large tailing dam carried 8 per cent. of the rock crushed in suspension. As the water on these fields does not run to waste, but is used over and over again in the batteries, there is really no loss incurred, as the suspended slimes eventually settle, the water only retaining a certain constant portion.

To give a further illustration of the effects of stamping on

the Witwatersrand conglomerates, I will furnish some figures communicated to me by Mr. Bettel, a gentleman whose name has also been closely connected with the cyanide process.

40 lbs. of tailings were caught at a battery in a tub, and at least 30 per cent. passed away as slimes, the ore coming through a 900 mesh screen. After drying, it was sifted through a screen of 1,600 mesh per square inch, and there remained on the sieve 1.85 per cent. (1)

It passed afterwards through three sieves as follows:

3,600 me	esh and t	here rer	nained o	on the siev	re 27.93 pe	r cen	t. (2)
7,225	,,	,,	,,	,,	20.74	,,	(3)
14,400	,,	,,	,,	,,	7.70	,,	(4)

The sands passing the last sieve were panned, and

Each of these grades was assayed, with the following results—

These figures are very instructive, as they show how very fine the ore is crushed in a battery, and that the material, after passing through a 120 mesh sieve per linear inch, can be concentrated, and will yield a concentrate of over double the value of the sands washed away. No doubt the practical part of these figures will be very soon appreciated on these fields when concentration will receive closer attention than it does at present.

After this notice of the important part which the slimes

play in the metallurgy of the Rand gold fields, I will revert again to the planning of the works.

The Plant.—The main features of a cyanide plant are the settling vats, the filter vats, the solution storage tanks, and the precipitation-boxes.

The filter vats are made of timber, or they are brick vats lined with cement. At the Langlaagte Estate and Gold Mining Company, circular excavations were made in the rocky ground, lined with bricks and cemented, forming tanks 40 ft. in diameter and 10 ft. deep, each holding 400 tons of tailings.

Filter vats made of timber will last for years, as contact with cyanide solutions does not seem to destroy it.

I have not been able to obtain the cost of a plant constructed in masonry, but I should consider it more expensive than the timber plant. Where wooden tanks are in use they are placed in such a position that free access can be had to the bottoms in case of leakage, which is an advantage.

The number of filtering vats required for a plant depends on the capacity of the battery, and the time it takes to treat a charge of ore. If we want to treat 100 tons of tailings daily, and it takes, say, 3 days to fill, leach, and discharge a vat, it will require 4 leaching vats of 100 tons capacity each (dimensions 22 ft. in diameter and 5 ft. high), but for safety one extra tank is added. The tendency on these fields is to construct a few large vats for the plant, instead of a large number of small ones. As long as shallow vats were employed, there was no difficulty in shovelling out the tailings over the sides; but with deep vats the bottom discharge was introduced by Mr. Butters. Before the tailings go to the leaching tanks they have to be freed of their slimes, and I shall now describe the two methods adopted for this purpose.

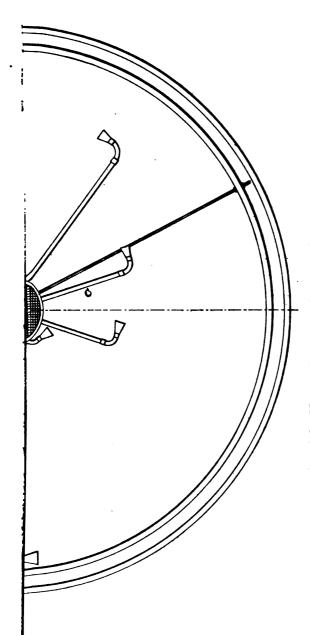
Intermediate Filling. Messrs. Butters and Mein's Distributors.—The first attempts at intermediate filling were made by running the battery tailings to the centre of a circular

vat, and allowing the overflow to take place at one point. This did not prove successful, because the sand piled up in a central conical heap, and the slimes settled in the deeper water around the sides of the tank. The next plan was to run the pulp into the vat through a series of stationary launders, delivering at several fixed points. This method improved the distribution, but the result was still unsatisfactory. Then, in order to give an uniform overflow at every point of the periphery of the vat, a circular trough was fixed round the top to collect the overflow and deliver it to a launder.

Each of these alterations was a step in the right direction, but the system of settling could not be considered successful until after the introduction of an automatic revolving distributor. This appliance consists of a central casting, with a vertical spindle A revolving in a footstep bearing B, which casting carried a conical hopper E and a number of radial pipes c with bent ends, as shown in section in Figs. 2, 3 (Plate II.), and 4 and 5.

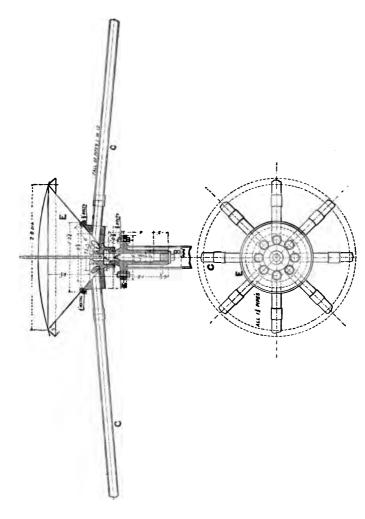
The distributor is fixed on an iron column in the centre of the vat. The bends at the end of the pipes cause the apparatus to revolve by the reaction of the pulp as it leaves the pipes. Each pipe has a different length, in order to distribute over a number of concentric circles. This also has its faults, as it was found that the slimes collected in narrow rings between the outlets of each pipe, giving rings of clean sand alternately with rings of slime. The difficulty was overcome by attaching flattened nozzles to the ends of the pipes, causing the pulp to spread over a wider area, and also by increasing the number of pipes.

As is noticed by this plan, the arrangement is a hemispherical bowl from which radiate 8, 12, to 16 pieces of pipes of various length, which is set in motion by the centrifugal action of the discharging water, something similar to a garden sprinkler, only that it revolves slowly. The bowl is covered with a coarse screen so as to prevent chips or leaves to enter and choke the pipes. The diameter of the discharge pipes is  $1\frac{1}{2}$  to  $2\frac{1}{2}$  in.



Figs. 2 and 3.—Messrs. Butters and Mein's Automatic Distributor.





Figs. 4 And 5.-Messes. Butters and Mein's Automatic Distributor (Size A).

It is necessary to fill the vat with clean water before admitting the pulp. If this is not done, the water is practically stationary, and a constant settlement of slimes takes place until the vat is full and the overflow begins, in which case the tailings in the lower part of the vat will always be more slimy than those in the upper part. For the same reason it is essential that the overflow be continuous until the vat is full of sand; for if any stoppage takes place slime settlement in excess occurs, and a complete layer of slime is formed across the vat which prevents the overlying sand from draining dry. Therefore, when the battery is stopped, an equal quantity of water should be supplied to the vat. When the pulp is admitted into the tank previously filled with water. the light slime remains in suspension and overflows into the annular ring which surrounds the tank at the top, and from the discharge opening is carried by a launder to the slime-pit.

When the vat is filled with tailings, the outlet pipe below the filter is opened and the water allowed to drain off, the draining taking about fifteen to twenty-four hours. When holes are dug down to the discharge doors, water again commences to flow from the outlet, consequently it is advantageous to dig these holes about six hours before the discharging.

One very important matter is the proper size of vat to be used for a given tonnage crushed in the battery. It is, of course, desirable to catch as large a quantity of slimes with the sands in the tailings as is possible without rendering the product unleachable. When the vats are too small they carry away too much fine sand with the slime; and if they are too large they catch too much slime, which settles in excess. The great difficulty to overcome yet with these intermediate vats is to get the last foot or two near the bottom properly drained, and if discharged and transferred to the leaching tanks in this wet condition, the excess of moisture dilutes the cyanide solution.

To facilitate and hasten the leaching, various devices have been adopted. At the Princess works, where the ground is steep, the drainage pipe has been extended down to the reserSLIMES.

voir, thereby causing a natural suction. At the Simmer and Jack works the drainage pipe is connected with a steam exhaust acting like an ejector, so as to cause a vacuum below the filter, and thereby the rate of leaching is increased. At the Worcester works the vats catch from the crushed ore from 75 to 80 per cent. of good leachable tailings, containing 12 per cent. moisture after draining eighteen to twenty-four hours.

The following are the sizes of the intermediate vats erected at some of the works:—

Meyer and Charlton Gold Mining Company, treating 120 tons per day, has 4 vats, each 20 ft. in diameter and 8 foot staves.

Pioneer Gold Mining Company, treating 70 tons daily, has 2 vats, each 20 ft. in diameter, and 14 foot staves.

Worcester Gold Mining Company, treating 70 tons daily, has 2 vats, each 20 ft. in diameter, and 8 foot staves.

Princess Gold Mining Company, treating 85 tons daily, has 2 vats, each 20 ft. in diameter, and 7 foot staves.

The Robinson Gold Mining Company, treating 330 tons per day, has 6 vats, each 24 ft. in diameter, and 11 foot staves.

When all the pulp is running into 1 vat, only about 66 per cent. of the crushed ore is caught, but the whole of this is clean sand and drains sufficiently. If, however, the total pulp from the battery was run into 2 vats, about 80 per cent. of the crushed ore, instead of 66 per cent., would be obtained from the distributing tank. After the water has been leached out, the ore is discharged through bottom discharges into trucks and taken to the leaching tanks. In some localities the distributing tanks are on a higher level than the leaching tanks, and the trucks are then run by gravitation to the leaching tanks. At some works the distributing tanks are at lower level than the leaching tanks, and then the trucks have to be hauled up by steam power.

The framework of the tram lines on which the trucks are hauled up to the leaching tanks rests inside the tanks and on the masonry foundation, and at large works there is generally a

double line of rails on top of the tanks. The vats and storage tanks are in the open, and not covered by a building.

The following are the advantages of intermediate filling, as introduced by Mr. Charles Butters \*:—

- 1. It is claimed that, by means of Mr. Butters' distributor, from 75 to 80 per cent. of sands, both coarse and fine, with some slimes, are collected in the intermediate tanks, the bulk of the slimes escaping with the effluent water, which is practically free from sands.
- 2. The water is drained off as near as possible, and when the intermediate vat is discharged through the bottom discharges, the sands during the operation get thoroughly mixed up, thus being in the best condition for treatment by cyanide.
- 3. Oxidation of pyrites is very slight, so that very little cyanide will be consumed.

To an impartial observer it would appear that the system of intermediate filling would commend itself as the one which is more practical, as the tailings undergo, so to say, a special preparation for the subsequent lixiviation. The expense of transferring the tailings from the intermediate tank to the leaching tank is so slight that it cannot be considered as an important item.

The cost of charging tailings and discharging the residues has been brought down at the Robinson Mine to 10d. per ton of 2,000 lbs., and generally stands in the accounts of other works at about 1s.

Messrs. Butters & Mein's Distributors are constructed in three sizes according to the following particulars:—

Size A Distributor has 8 distributing pipes, all of 1½ in. diameter, and is the size of distributor used on batteries up to 30 stamps.

\* The position held by this gentleman in the Witwatersrand gold fields is so influential, that the accompanying portrait will be of great interest to many readers of this book; and I have great pleasure in inserting it here, if only to emphasise my sense of the acknowledgment due to Mr. Butters in regard to the information and assistance so freely afforded to me when collecting materials for this volume.—M. E.



MR. CHARLES BUTTERS.

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Size B has 12 distributing pipes—6 of 2 in. diameter, 2 of 1½ in. diameter, and 4 of 1½ in. diameter—and is the size of distributor used on batteries of from 30 to 70 stamps.

Size C has 16 distributing pipes—2 of 2½ in. diameter, and 14 of 2 in. diameter—and is the size of distributor used on batteries of from 60 to 120 stamps.

The above sizes of distributors have been calculated on the average crushings per stamp for the Rand.

Direct Filling.—This method, introduced at the works of the Heriot, City and Suburban, Crown Reef, Paarl Central, and Geldenhuis Estate companies, consists in passing the pulp as it leaves the plates into a hydraulic separator, a kind of crude spitzlutte. The pulp is here divided into two streams, one overflowing, carrying slimes with very fine sands; the other, consisting of coarse sands, some fine sands and slimes, which are conveyed by means of an india-rubber hose to the leaching tanks, in which one or more Kaffirs are employed to effect the even distribution of the pulp, by moving the hose about to different parts of the vat. The water passes off by adjustable gates fitted inside the vats, carrying with it fine sands, slimes, and some coarse sands. The advantages of the process are:—

- 1. This method treats pyritic tailings with the minimum of oxidation, as they are not exposed to the action of the air from the time they leave the battery.
- 2. A second handling of the tailings before treatment is avoided.
- 3. A preliminary rough concentration, or rather classification of the coarser particles of the tailings is effected.

There is at present a great controversy going on regarding the advantages of direct filling as against intermediate filling, and according to Mr. Bettel the disadvantages of the process are:—

"1. The tailings pack tightly in the vat, and consequently do not drain completely, and a diffusion of the first cyanide

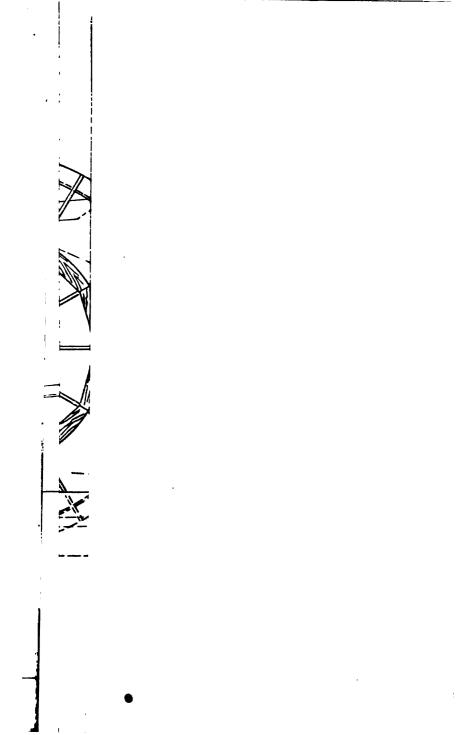
solution which is applied takes place at the commencement of leaching, causing loss of cyanide and gold. At the Crown Reef works I noticed that the distribution seemed to be pretty regular, and drainage can be assisted by means of exhaust pumps.

- "2. The distribution of the sands and slimes is not so even, and some sands escape treatment, being protected by impervious layers of slime, the cyanide naturally escaping by the paths of least resistance. In leaching tanks where an uneven distribution of slimes and sands takes place, the slimy portion will not drain off; and on discharging such a tank, it is easily noticed that the streaks of slime are saturated with moisture and are still gold bearing, whereas the sandy portion has the solution drained off. The importance of an even distribution and mixture of the pulp can hardly be estimated.
- "3. At most of the works where direct filling is introduced, square cement tanks are employed, and the discharging of these is not so practical as the wooden ones fitted with bottom discharges."

In Plate III. (Fig. 6), the tailing wheel, vanner room, and cyanide vats at Jumpers' mine are shown in section.

The Filter or Leaching Tanks.—These are, in most instances, made circular, that form being the strongest. They are from 20 to 42 ft. in diameter, and from 8 to 14 ft. in height, and should be constructed of well-seasoned lumber, with staves 3 to 4 in. thick, having their inner and outer faces cut to correspond to the arc of circle of the tank, and their edges radial to this circle (Fig. 7). The staves are not tongued or grooved, the pressure of the hoops being sufficient, if the tank is well made, to make them perfectly tight. The staves should be at least 1 ft. longer than the inside depth of the tank, and gained from 1½ in. into the bottom timbers, with a chime of several inches.

The bottoms are made of 3 by 9 in. deals, tongued and grooved (Fig. 8), and put together with white lead, or litharge and glycerine. The hoops should be made by wrought iron



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rods from  $\frac{3}{4}$  to  $1\frac{1}{2}$  in. in diameter, according to the size of the tank, with threaded ends passing through wrought iron lugs and tightened by hexagonal nuts. When the tanks are of large diameter these hoops are made in sections. The outside of the tanks can be painted in lead paint.

The bottoms of the vats rest on wooden beams 6 by 9 in., placed 18 in. apart. These beams are placed across the stone

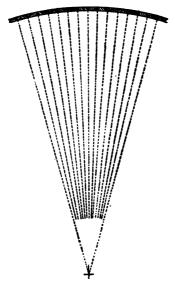


FIG. 7.-STAVES CUT TO CIRCLE.

foundation, and rest in their turn on planks 13 by 11 in. The planks are put between the stone foundation and the beams to merely ensure a perfectly level surface.

The construction of these vats should not be entrusted to the hands of any other workmen than experienced coopers.

It is obvious that tanks holding such enormous weights should rest on good foundations, and in every case where

wooden foundations have been used the result has been that the tanks settled, got out of plumb, and leakages occurred.

The filters are constructed of wooden slats, 1½ by 4 in., set 12 in. apart, fastened to the bottom by wooden pins. Grooves ¾ in. deep and 3 in. wide are cut in a number of places in the bottom of these slats to allow a free passage of the solution along the bottom. On top of these slats are laid strips

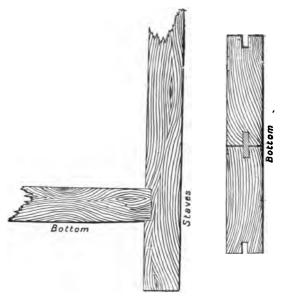


FIG. 8.-CONSTRUCTION OF FILTER VATS.

of wood I by I in., only I in. apart from each other, making openings I in. square. Between the ends of this wooden grating and the inside of the tank an annular space of about  $1\frac{1}{2}$  in. wide is left, which is partly filled by a strip of wood I in. thick, bent to the circle of the tank. Over this and the slats is placed cocoa-nut matting and burlap, and held by a rope  $\frac{1}{2}$  in. in diameter, which is driven into the space remaining between

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the strips of wood and the staves of the tank. On top of the matting are laid again slats of wood 1 by 3 in., parallel one to the other, about 6 in. apart, their object being to protect the matting from being injured when shovelling the tailings through the man-holes into the trucks below.

The stone foundations are usually 6 ft. 6 in. high above the level of the rails, and are composed of a series of walls closed at their ends, leaving one or two passages underneath for the trucks (Fig. 9).

Each leaching vat has a separate drain pipe, 1 to 2 in. in diameter, and these pipes are so arranged in the extractor house, as to lead the strong solution to the strong extractor box

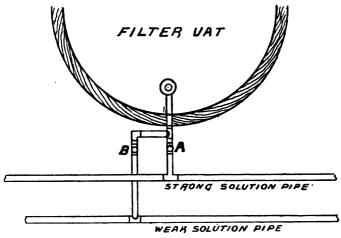


Fig. 10.-Solution Pipes.

and the weak solution to the weak extractor box. In some works there is one main collecting drain pipe for strong solution and one for weak solution, and the connections are shown in Fig. 10. By shutting the valve, A, leading to strong collecting pipe, and opening valve, B, leading to weak collecting pipe,

the flow is regulated. Filtration is best assisted by causing a vacuum under the filter bed; by connecting the drain pipe with a steam pipe and passing a jet of steam through the same, a vacuum is created under the filter bed.

I should also mention that the best and cheapest method of discharging the tailings from the leaching vats is to sluice them out from a side door, but for this purpose a stream of running water is required, which on these fields is not available.

Mr. Feldtmann \* describes a system of discharging tailings from the leaching vats through a bottom discharge door into a launder, whence a copious stream of water carries the residues into the creek below.

The discharge doors can also be made on the side of the vat when the residues are to be sluiced out.

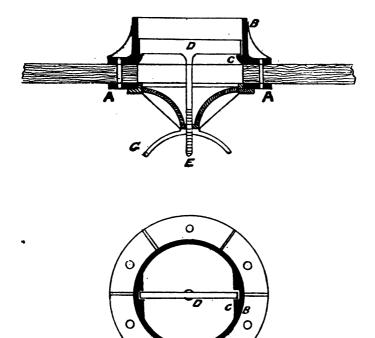
The round wooden filter vats on these fields are discharged by bottom discharge doors, which are closed by means of Butters' discharge lids. According to the size of the vats, there are two, four, six, or eight of these discharge openings to each vat.

Figs. 11, 12 show the arrangement. On the bottom side of the tank a cast-iron ring, A, is bolted to the cast-iron cylinder, B, inside the tank. Inside the cylinder is the projecting lug, c, upon which rests the hanger, D, which forms part of the screw, E; the cast-iron cover, when placed in position, is simply fastened by the nut, G, and, screwing the same firmly, the whole arrangement becomes water-tight. The faces of the ring and the cover should be planed, so as to make a good joint. There are other methods of closing the discharge openings. When a tank is to be filled, a clay luting is given inside the iron cylinder, and then the same is rammed full with tailings. When filling the tank with tailings, especially into deep vats, a length of 3 to 4 ft. pipe is put over the discharge holes, and then the tailings are dumped in. It will be easily understood that in discharging a deep tank it facilitates the running of the tailings into an outlet if the same is within a few feet of the surface,

\* "Notes on Gold Extraction." Argus Printing and Publishing Co., Johannesburg, 1894.

instead of having to push them down 13 or 14 ft. by means of long poles.

The cocks and valves should be of iron.



Figs. 11, 12.—Butters' Discharge Lid (scale 1 in. = 1 ft.).

Pumps.—Several varieties of pumps are used to raise the solution from the sumps to the lixiviation tanks, and to provide circulation if needed. Centrifugal pumps are mostly used on these fields.

Stock Solution Tanks.—There are generally three solu-

tion tanks at each plant, built very much the same as the leaching tanks, with the exception that they have no filters, man-holes, &c. They are of different capacity, according to the size of the works, and are required to be of sufficient dimensions to store enough solution to keep the works going, without having to run any to waste.

Inside the tanks are gauges indicating the volume of solution. The stock solution tanks are usually 20 ft. in diameter, and from 7 to 14 ft. in height. One is for strong, one for weak solution, and one for alkaline wash. Every foot in height in a 20-ft. tank represents 10 tons of solution of 2,000 lbs. per ton.

To calculate the cubic contents of a circular tank, the following formula is employed. Multiply the square of the radius 10<sup>2</sup>) with 3.14, and the product by the height of tank (6 ft.).

$$10^2 \times 3.14 \times 6 = 1884$$
 cubic feet.

1 cubic foot of water weighs 62.3 lbs. Therefore,

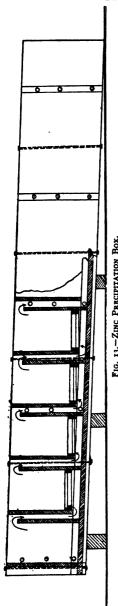
$$1884 \times 62.3 = 117573 : 2000 = 58.78 \text{ tons of water.}$$

If we desire to prepare a 0.3 per cent. stock solution,

 $\frac{117573^{2} \times 0^{\circ}3}{100} = 352^{\circ}71$  lbs. of cyanide will have to be dissolved in it, making allowance for any impurity in the cyanide. After treatment we find that the solution analyses only 0.16 per cent. Consequently, by multiplying

 $\frac{117573.2 \times 0.16}{100} = 188.11$  lbs. is left in the solution; and to make the same up to 0.3 per cent., another 164.61 lbs. of cyanide will have to be added.

Zinc Precipitation Boxes.—These are made of 1 to 2 inboards, and are oblong boxes of various dimensions, which have to be in proportion to the quantity of solution which passes through them. In large works, the boxes are 20 ft. or more in length, 3 ft. high, and 3 to 4 ft. wide. There are separate boxes for the strong and for the weak solutions to pass through. At most works there are four of these boxes, placed in what



is called the extractor-house, which also contains the machinery, pumps, furnaces, &c.

The precipitation-box is divided into several compartments by partitions and baffle boards, in such a way that the solution is forced to flow upward through the zinc shavings, which are held in trays several inches above the bottom of the troughs. Fig. 13 shows the construction of the troughs.

The first division has not got any zinc shavings in the same, as here the solution enters, and any sediment or fine slime which may have passed through the filter settles here. If any intermediary settling tanks are used, as at the Worcester works, this first compartment can be utilised also to hold zinc shavings. From the first compartment the solution flows over the partition, and then down the space, and upward through the trav holding the zinc shavings. The baffleboard is held in place in the position shown in the drawing by being nailed fast to the sides, and reaches a few inches above the level of the solution. From this explanation it becomes clear that the solution has to pass its downward and upward course till the last partition is reached, and from here passes through a pipe to the collecting sump or tank.

The zinc box compartments are fitted with removable trays, made of

wooden frames supporting wire screen of  $\frac{1}{8}$ -in. mesh. The gold in the solution settles on the zinc as a brown coating, and which soon, as it accumulates in a finely powdered state, falls through the screens to the bottom of the trough. In the last partition of each box there is no zinc, but the tray here is utilised to hold cyanide of potassium in lumps to make up its standard strength before pumping the same into the storage tanks.

Over the zinc there is placed a light wooden grating, and the whole trough can be covered by a strong wire netting to secure against theft, as the same can be kept under lock and key.

At some works a wooden launder, covered with a lid and also under lock and key, is attached longitudinally to the box, and from each compartment in the trough when a clean up is made, a plug is withdrawn and the slimes which have accumulated in the bottom are washed through the launder on to a filter and collected. In most works, the clean-up is made differently, as described later on. The zinc trays rest on cleats, several inches above the bottom, and have handles on the sides so that they can be easily lifted out when cleaning up. After passing through the precipitation-boxes the solution is pumped back to the storage tanks, and is used continuously and not run to waste. The dissolved zinc does not accumulate in the stock solution to a great degree, and is probably precipitated in the lixiviation tanks with the charges of fresh ore.

### CHAPTER II.

#### EXTRACTION BY CYANIDE.

Synopsis of the Process.—On the Witwatersrand gold fields, the cyanide process has been mainly adapted to the re-treatment of the tailings. As the largest proportion—and amongst it the coarser particles of gold—have been extracted by previous plate amalgamation, the precious metal in the tailings is in a very fine state of division, and therefore amenable to cyanide treatment. It must be here remarked that the coarser the gold, the longer it takes to dissolve it; and it is recommendable, therefore, that all ores should be first submitted to plate amalgamation before submitting them to the cyanide treatment.

When the first cyanide works were erected on these fields, the old accumulated stocks of tailings had to be dealt with. Owing to their long exposure to the atmosphere, changes had taken place in their chemical composition, which caused at the onset some difficulties, but these were soon overcome on the application of the proper remedies. The ores which came from the upper levels, or the oxidized zone, always carry a small proportion of iron pyrite, which, on exposure, becomes oxidized. It is only when free milling ore tailings are taken directly from the battery to the cyanide works that they do not

contain any decomposition products and are in a proper condition for cyanide treatment.

The course of the treatment in its successive stages may be thus summarised:—

First Stage.—Passing an Alkaline Solution or caustic wash through the ore to the point of saturation, as the tailings always contain a certain amount of organic matter, acid salts, &c. By running on this alkaline solution, a considerable saving is effected in the consumption of cyanide of potassium, and the tailings get the advantage afterwards of the full strength of the strong solution. Strength of this first alkaline solution may be 0.15 per cent. KCy., and may contain 4 ozs. of caustic soda per ton of solution. Caustic soda dissolves out organic matter. Excess of lime destroys cyanide.

Second Stage.—The Strong Cyanide Solution.—This solution varies in strength from 0.3 to 0.5 per cent. in KCy. to suit the richness and nature of the tailings under treatment, and the proportion of solution to be run on should not be less than one-third the weight of tailings in the vats. When this solution is run on, though the first solution has drained out, the tailings still contain a considerable quantity of the former solution. This should be displaced by allowing the second solution, or strong solution, to drain down immediately the vats are filled with the strong solution, for say about two hours (according to the capacity of vats and nature of tailings), and when this has been displaced, sufficient strong solution should be run on to make up the required proportional amount.

The vat is now full of strong solution, which, in some cases, may be leached out immediately, though generally it is advisable to leave it in contact with the mass for a short period—say, three hours—to give the solution time to penetrate any lumps.

When this solution is leached out, or only a very small stream is coming away, it should be allowed to continue draining about four hours longer. During these last four hours air is taking the place of the solution, and the gold is in con-

tact with a strong solution of KCy., in presence of oxygen, which produces a more rapid and effectual dissolving of the gold.

For proof: Take two watch glasses, fill both with cyanide solution of the same strength. In one, place a piece of gold leaf on the surface of the solution, while in the other immerse the gold under the solution. The gold leaf on the surface in presence with air disappears rapidly and is completely destroyed, whilst the other dissolves very slowly.

Third Stage.—The Weak Solution.—After the strong solution has been run on, we may safely assume that most of the gold is now in solution, and the object of the third operation is to wash out the dissolved gold. Therefore, after the strong solution has been drained out, sufficient weak solution containing 0.15 per cent. KCy. is run on till the total quantity of solution and washes represent 75 to 80 per cent. of the weight of the ore.

Fourth Stage.—The Water Wash.—After the weak solution, water wash is applied, and the quantity so applied is not to be less than 7 per cent. of the weight of the ore, and, indeed, more is necessary.

The Precipitation.—The cyanide solutions containing gold, as they flow from the leaching vats, are passed through one or more precipitation boxes. On the zinc shavings the gold is precipitated, and the quantity of solution flowing through each box must be properly regulated. The only satisfactory method of knowing if proper precipitation is taking place is by having the solution regularly assayed. This is an important point, as good results cannot be obtained if the solutions leaving the precipitation boxes are rich in gold.

It is essential for the man on the shift to know the strength of the solution at the different stages of leaching: first, that he may know into which one of the precipitation boxes the solution is to be passed; and then he must ascertain the strength of the solution leaving the precipitation-boxes, in order to let them flow into their respective storage tanks.

The gold in solution increases and decreases in quantity much in the same way and same time as the cyanide in solution increases or decreases. In the precipitation of the gold in the boxes, it is important to keep a sufficient stock of zinc shavings in each compartment, and so to regulate the flow of solutions as not to incur danger of fine gold precipitate being carried away. The zinc shavings are prepared by turning down zinc discs on a lathe. The discs for this purpose may be cut out of No. 15 gauge metal, and may measure from 6 to 12 in. in diameter, a hole being punched in the centre for the mandril. It is usual to put bundles of twenty such discs on a mandril. The speed of the lathe may be anywhere from 150 to 350 revolutions per minute, and the shavings are turned off by hand with an ordinary carpenters' mortice chisel.

Precipitation of the gold varies somewhat with different classes of ore treated. The completeness of the precipitation appears to depend in a measure on a slight excess of cyanide of potassium being present in the solutions. Roughly speaking, it may be said that if solutions leaving the zinc boxes assay more than 2 dwts. per ton, the precipitation is not as it should be. This may be owing to the paucity of zinc in the boxes, which should be instantly rectified; or to too great a speed in the flow of the solution; or, in very exceptional cases, to insufficient cyanide in the solutions.

In the case of some Lydenburg cupriferous ores, it was found that, by making the solutions up to working strength before passing them through the zinc boxes, the result was that solutions which had before assayed several ounces were reduced to a few dwts. per ton.\*

The zinc shavings in the boxes may require replenishing every day to replace the amount consumed, or they may run a week at a time without requiring replenishing.

Having, by passage of the solutions through the zinc shavings, reconverted the gold into the metallic form, the

<sup>\*</sup> According to Mr. Feldtmann.

process of collecting the metal and putting it into ingots will be another stage of operations, and this will be described presently under the clean-up.

The exposure of the solution to the zinc after the gold is precipitated can only result in a loss of zinc, and—what is of much greater importance—unnecessary decomposition and loss of cyanide.

The precipitation boxes are set at a slight grade, as shown in the plan. Mr. Philip Argall, M.R.I.A., says that no iron or metal, other than zinc, should be exposed to the solutions in boxes; even the iron wire screens, used to support the filiform zinc, have been shown to cause an unnecessary consumption of zinc and cyanide, through the electro-chemical action induced between the metals.

Conditions which Influence Precipitation.—Mr. W. Bettel states that in the precipitation of various metals from cyanide solutions by zinc, the precipitation is influenced by the following conditions:—

- (a) Different strengths of solvents.
- (b) Impurities in solutions (which may be derived from ores) affecting precipitation.
- (c) Influence of rate of flow of solution through extractors,—past a given area of zinc—in gallons per minute.
- (d) Amount of gold in solution to be precipitated, and maximum precipitation from solutions of various richness.
- (e) Effect of alkalinity, acidity, or neutrality of liquors on rate and percentage of precipitation of metals from cyanide solutions.
- (f) Influence on the rate of precipitation of gold of such inert substances as carbonate of lime, clay, oxide of zinc, and coating of zinc, &c.
- (g) Other physical conditions of zinc, affecting precipitation of metals from cyanide solution.

Time taken by Treatment on Clean Tailings.—The method of treatment described below has been adopted at the Rand Central Ore Reduction Company's works, the figures

having been kindly supplied to me by the working chemist, Mr. Blomfield :-

	Alkaline W Filling with Caustic	0.16							time	e 2 l	nours.
]	Leaching.	•	•		•		•	•	,,	3	,,
2nd.	Strong Sol	ution.									
1	Filling with	0.32 I	<b>ζ</b> Cy.	solut	ion a	ınd c	ontact		,,	5	,,
1	Leaching.		•		•	•			,,	8	,,
1	Leaching dr	у.	•	•	•	•	•	٠	,,	4	,,
3rd.	Weak Solu	tion.									
	4 washes at	1 hour	each	١.			•		,,	4	,,
•	4 washes at 4 leachings	at 4 ho	urs e	ach		•		•	,,	16	,,
	Water Wa										
	Filling .								,,	I	,,
	Filling . Leaching .	•		•	•	•	•	•	,,	7	,,
										_	hours.

This treatment was applied to 7½ dwts. tailings, residues being from 15 grs. to 1 dwt. per ton; 3 lb. of cyanide being the consumption per ton of ore.

On these fields it has been found that 12 to 24 hours' contact with the strong solution is sufficient to effect a solution of the gold in the tailings. The subsequent treatment with weak solutions has the object to displace the gold solution and to wash out the already dissolved gold. The displacing of the weak solution by a final water wash tends to reduce the loss of dissolved gold to a minimum, as it displaces the weak gold solution, and the gold which is left in the tailings is generally that still retained in the coarser quartz particles, from which it has not been liberated by stamping, and whatever gold has remained undissolved in the iron pyrites.

The time of treatment of each vat varies, and takes from 50 to 140 hours, according to circumstances, and the size of the tanks employed.

The Clean-up.—In making the clean-up, the tray holding

the zinc shavings is lifted out from the last compartment, and pulsated up and down in the solution, so that the fine particles of slime and zinc fall through the sieve and settle in the bottom of the box. Before starting this operation, it is advisable to run into the box a sufficient amount of clean water to remove the cyanide solution, as the latter is injurious to the workmen. The Kaffirs, however, put their arms and hands into it without being affected. The zinc shavings are taken out of the tray, which is placed on a rack above the box, so as to allow solution to drain back into the box. They are also rubbed in the water, to remove, as much as possible, all gold adhering to them.

The tray is turned over and brushed down so as to remove any gold adhering to it. The zinc shavings get very hot on account of oxidation, and it will be noticed that steam arises from them. They should be exposed, therefore, as little as possible to the air. The solution of water in the zinc boxes is pumped into settling tanks, where it is allowed to stand for two weeks, so as to give the extremely fine particles of gold which are held in suspension time to settle. In pumping out the precipitation boxes, great care should be taken not to disturb the gold zinc slimes in the bottom. With this aim, the water is pumped out to within 2 in. of the slimes, and the indiarubber suction hose moved into the second compartment, and so on.

The slimes are then pushed back with a scoop to one corner, and the supernated liquor allowed to stand for a while and pumped away again into the settling tanks. The slimes are now scooped into enamelled iron buckets and discharged on to a fine sieve—say 900 mesh—and washed and rubbed into the gold clean-up tank.

After the water is settled in the clean-up tank it is syphoned or pumped off, and the precipitate (called gold slimes) is drawn off through the plug holes on to a calico or linen filter, or into a filter press.

The zinc shavings, some of which will be found to have quite a brown coating of gold, which cannot be removed even

by rubbing, are again returned to the precipitation-boxes, and fresh zinc put on top. The gold which sticks to the zinc will be recovered in the next clean-up. After the gold slimes are sufficiently dry to be handled with a scoop, they are dried on an iron plate or on iron pots, and are then ready for roasting and smelting.

The object of the roasting is to oxidize the greater portion of the zinc which, in the form of small chips and shavings, has fallen through the zinc box trays, so as to cause it to combine in the subsequent smelting with the fluxes and leave the bullion fairly fine. Oxidation by the aid of atmospheric air is sufficient, but a certain amount of the zinc oxide subsequently becomes reduced by the carbon of the plumbago melting pots, and re-enters the bullion.

A good method of roasting has been found to be the addition of a little nitre, say about 3 to 10 per cent. to the precipitate. Mr. Feldtmann suggests that it is best applied as a strong solution before drying the precipitate, so that it gets equally mixed with the whole mass. In the subsequent roasting, the nitre not only assists by yielding up oxygen to the zinc, but to some extent also appears to flux the zinc oxide, forming zincate of potash, which is not so readily reduced as zinc oxide. Where the precipitate is very sandy—owing to tailings coming through the filters—nitre roasting is not so successful, as it tends to cake. By the addition of nitre the tendency of the precipitate to dust on stirring up in the roasting furnace is minimised, the amount of flux required in smelting is reduced, and the resulting bullion is better.

In roasting the precipitate care should be taken not to raise the temperature much above a dull red heat (to avoid partially fusing it to a pasty mass), and not to stir too violently, especially just at the commencement of the roast, or dusting and consequent loss is the result.

Mr. Butters, at the Rand Central works, has a muffle-roasting furnace in which to dry and roast the slimes. The bottom of the furnace consists of a cast-iron pan, and the wet slimes are charged on to this pan, and, when dry, a damper is closed, which turns the flame through an opening in the fire-bridge, under the iron pan, and the slimes are carefully stirred to avoid dusting, and as much of the zinc as possible is driven off during the roasting.

The clean-up is generally made once a month.

Smelting of the Slimes.—The dried precipitate is now ready for the smelting process, and graphite crucibles are employed for the operation. The fluxes commonly used are bi-carbonate of soda, borax, and sand.

Examples of various fusing mixtures are given below, but it should be well understood that any one of the fluxes may have to be increased or decreased according to the amount of impurities present:—

	<ol> <li>Clean Precipitate.</li> </ol>	<ol> <li>Very zincy Precipitate.</li> </ol>	<ol> <li>Very sandy * Precipitate.</li> </ol>			
Precipitate .	30 lbs.	30 lbs.		30 lbs.		
Bi-carbonate soda	15 ,,	15 "		20 ,,		
Borax	8 ,,	12 ,,		10 ,,		
Sand	5 ,,	5 ,,		_		
Fluor spar .	<del></del> .	_		2 ,,		

Precipitate and fluxes are well mixed and charged into the plumbago crucibles. The smelting furnaces, which may be constructed to take two or three pots at a time, should be good ones, as the heat required for this first fusion is rather in excess of the ordinary gold melting temperature. After the charges in the pots are run down, more of the mixture may be added from time to time—the whole of a charge, as given above, will go into two No. 35 crucibles—and everything being fused until perfectly liquid, the contents of the pots are poured into moulds. Conical-shaped moulds are the best suited for this work. The metal settles to the bottom, and, after cooling, may be turned out and freed from the slag by breaking off the latter with a hammer. The slags, which contain a large percentage of silicate of zinc and soda, corrode the pots, and during smelting there is a heavy evolution of fumes of zinc

<sup>\*</sup> According to Mr. Feldtmann's formula.

oxide, causing, most likely, losses of gold. At the Robinson Mine condensing flues connect the furnaces with the chimney.

The several pieces of bullion thus obtained at one clean-up, are, consequently, remelted with borax and run together into one ingot. This remelting should be done at as low a temperature as possible, so that the metal may solidify almost as soon as it is in the mould, otherwise liquation results, and it becomes exceedingly difficult to obtain anything like a representative sample of the bullion for assay. The slags, which generally contain a considerable amount of gold in beads, are crushed up and panned, or cradled, to obtain the metal.

The slags from this operation are difficult to re-smelt, which may be owing to the presence of carbon contained in the zinc. At the Rand Central ore reduction works, Mr. Butters is erecting a small matting furnace, where he intends to smelt the slags with copper ore concentrates and collect the gold in a matte.

The fineness of the bullion resulting from the cyanide process ranges from 600 to 800.

Besides gold, silver, and zinc, there is some lead in the ingots, as the zinc employed contains a certain per cent. of this metal, and also carbon, which also is found in the ingots.

The zinc supplied by the Vieille Montagne Company is the best and purest for the purposes of the process.

The Treatment of Acid Ores or Tailings.—Under acid tailings are understood those tailings which contain the decomposition products of the iron pyrites. These products consist chiefly of free sulphuric acid and soluble metallic salts, such as proto-sulphate or per-sulphate of iron, or insoluble basic iron salts. All these substances are destructive to cyanide, forming with it compounds useless in the extraction of gold. The reactions which take place, when these salts are in the tailings, will be found explained in a subsequent chapter (Chapter IV.) on the chemistry of the process.

The ores on these fields at a comparatively shallow depth become very pyritic, but outside of iron pyrites the main reef series carries no other sulphur combinations, or, at least, in such small proportions, as to have no practical importance.

The treatment of acid tailings on these fields offers more difficulty than in the case of the "free ore tailings." The oxidation products of the pyrites have to be neutralized by the addition of alkalies or alkaline earths—either caustic soda or lime—with or without a preliminary water washing, to remove such soluble salts or "cyanicides" as may be present.

Caustic lime, in a powdered form, is generally added now on these fields. With very acid tailings—namely, pyritic tailings—which have been exposed for some time to the oxidizing influence of the atmosphere, as much as 21 lbs. per ton is added. With fresh tailings 1 lb. of lime per ton of tailings is sufficient. In some works the requisite quantity of lime is added to each car load of tailings as the same is hauled up to the leaching tanks. In other works 6 or 10 tons of tailings are dumped into the leaching tank, and the same levelled off and the lime sprinkled over it. The practice of putting all the lime on top of the tailings after the vat is filled, cannot be recommended: the lime forms a pasty mass, and the alkalinity does not penetrate through the total height of the ore, so that in deep tanks it will be found that the top layer, say for one-third of the height of the ore, will be neutralized, whereas two-thirds nearer the bottom will remain acid. The length of treatment varies, and later on in this volume (Chapter III.) I am giving the modus operandi of the process at some of the principal works on the Witwatersrand.

Concentration and Treatment of Concentrates.—The question whether it pays to concentrate the tailings before sending them to the cyanide works has not yet been definitely settled on the Rand gold fields. Considering that during the short period of treatment which the tailings undergo, the gold cannot all be extracted from the pyritic particles, it seems rational that a separation of the pyritic matter should be effected, and the concentrates treated separately.

The only mines where concentration previous to cyaniding

has been carried out for any length of time are the Crown Reef and the Langlaagte Estate and Langlaagte Block B. Mines, where the concentrates are treated separately with cyanide. At these mines I am informed that it pays to follow this method.

Mr. Williams, of the Crown Reef, collects his concentrates by a crude system of classification, which costs him 10d. per ton of concentrates, whereas at the Langlaagte Estate Frue vanners are employed. The manager there informs me that the cost of cyaniding the concentrates amounts to only 17s. per ton.

I believe that when the question has been thoroughly studied it will be found that a system of proper concentration before cyaniding will pay. Up to the present there is not a single mine on the Rand equipped with a proper concentrating plant, as such ores as exist there have to be classified before an attempt can be made to concentrate them. At the Langlaagte Estate, where 22,000 tons of ore are crushed monthly, they get on an average 350 tons of concentrates, or about 14 per cent.; this quantity could be easily doubled by the application of a proper plant. Most mills have Frue vanners, but this excellent machine cannot do the work alone on these ores. I may mention that Mr. Rademacher, engineer of the Humboldt works at Cologne, is now engaged at the Langlaagte Block B upon experiments with the view of concentrating the pyrites out of the slimy particles of ore. The results there will throw considerable light on the subject, and prove if I am right.

High extraction by cyanide can be obtained from concentrates, only the process does not seem to act alike everywhere on the concentrates, and possibly the physical nature of the pyrites is not alike in every mine. In some cases it may become necessary to grind the pyrites, so as to liberate the gold, and make it more amenable to cyanide. It is stated that gold contained in pyrites is extracted more easily when the concentrates are mixed with sands. Clean concentrates, therefore, are more difficult to treat, as owing to their high specific gravity they "pack" and resist percolation.

At the Crown Reef mine they collect by means of 3 spitzlutten, about 600 tons of concentrates a month, out of 17,000 tons of ore. A  $r\frac{1}{2}$ -in. pipe leads the heavy sands and pyrites to settling tanks 30 ft. in diameter and 6 ft. high, which assay 23 dwts. per ton. The main object of this rough concentration is to eliminate the coarse sands so as to submit them to a prolonged treatment with cyanide. The cost of this system of concentration is only £25 per month—namely, the pumping of the water which is needed for the hydraulic classifier—as they are worked by an ascending stream of water. The concentrates, or rather the classified material, by being collected in wooden vats, is constantly under water and does not get oxidized. The material collected is not very coarse, as in the battery they use goo-mesh screens.

From the storage tanks the material is taken to the leaching tanks, and submitted to the action of the solutions for 16 to 18 days. The strength of solutions employed is 0.25, 0.1, and 0.05 per cent. respectively; about 1 lb. of lime is added to each ton of concentrates. Strong solution is run on for 4 days or more, till its strength remains constant.

The residues from these concentrates assay  $t_{\frac{1}{2}}$  dwts., giving an extraction of 94 per cent. The consumption of cyanide with these concentrates is about  $t_{\frac{1}{2}}$  lbs. per ton. The weaker solutions are constantly circulated through the tanks till a high point of extraction is reached. The Witwatersrand ores carry from 3 to 4 per cent. of pyrites; and I believe the time of treatment could be lessened by employing a somewhat stronger solution.

At the Robinson mine 3 per cent. of the total weight of the ore is caught on Frue vanners, as concentrates assaying 4 to 5 oz. Extensive experiments have been made with cyanide, but the results were not satisfactory, and consequently the concentrates are treated at this mine by chlorination, which costs  $\pounds_3$  per ton.

The tailings at the Robinson are elevated by means of a bucket-wheel, and passed through intermediate tanks, fitted with Butters and Mein's distributor; there are six intermediate tanks, and 30 per cent. of the ore goes into the slime pits.

The tailings, which go to the cyanide works, assay 6½ dwts.

The slimes assay 5½ dwts.; and Captain Mein tells me that the residues, after cyanide treatment, assay only 1 dwt. This would mean a higher extraction than on any mine on the Rand. He finds that there is a discrepancy of 1 to 2 per cent. between the actual gold produced and that determined by assays.

Adverse Conditions which affect the Cyanide Treatment.\*—1. The action of impurities and base metals in ores prevents solution of the gold and effects the decomposition of cyanide. This will require a complete study by analysis of the component parts of the ore.

It is necessary to examine the condition of gold in residual tailings or concentrates from the cyanide treatment.

- (a) In this connection it has to be ascertained if a certain percentage of dissolved gold (the auro-potassic cyanide) has or has not been completely washed out.
- (b) Also if a certain per cent. of gold is still present in coarse particles capable of being amalgamated.
- (c) Also the percentage of gold encased in quartz, or pyrites, to which the solution has not had access.
- (d) Also the occurrence of gold in lumps of slime after treatment by cyanide; the gold supposed to be precipitated by the action of iron salts producing a ferrous ferricyanide from auro-potassic cyanide and free cyanide, thus explaining some low extractions from weathered ore and concentrates.

The presence of copper combinations is very detrimental to the success of the MacArthur-Forrest process. The ores of the Black Reef series, near Johannesburg, carry a small quantity of copper pyrites, and consequently the extraction of gold by cyanide does not give as good results as on the Main Reef series.

In this connection it is important to mention here a point which the inventors of the cyanide process have strongly emphasised—namely, the selective action of weak solutions of cyanide for gold in preference to other metals. It is a strange

<sup>\*</sup> According to Mr. Bettel.

fact that, while gold is one of the most difficult of metals to dissolve in acids, it is extremely susceptible to the weakest solution of cyanide, when copper and other metals might remain unaffected in it. The successful use of very weak solutions may have much wider and more important consequences.

It is well known that the MacArthur-Forrest process has failed in some mining districts in the treatment of complex ores; and it has seemed to me not unlikely that when experimenters with the cyanide process have met with difficulties such as this, they have increased the strength of their solutions, using more cyanide in the hope of overcoming the disturbing presence of the baser metals.

I shall presently give an account of the process introduced by Messrs. Siemens and Halske, who have adopted the diametrically opposite course; they have taken weaker solutions. If they had confined themselves to the chemical precipitation of the gold upon zinc or some other metal, as in the MacArthur-Forrest patent, they would have failed, because a point arrives at which chemical precipitation, in exceedingly dilute solutions, becomes imperfect. But the introduction by Messrs. Siemens and Halske of the agency of electricity, solved the problem at once. Now it is quite possible that, when the complex ores of other countries are treated with very weak solutions and by the electrolytical method, the weak solution will attack the gold in the ore, but may leave the other metals unattacked. There is a wide field open here for investigation and research.

The Practical Results.—Before we begin the treatment of the tailings, we meet with an acknowledged loss of 25 to 30 per cent. of the gold which goes into the slimes. It is true this gold is not lost, for it is still there. It is still an asset which accumulates monthly by the thousands of ounces in the slime-pits; but in the present state of our science and knowledge it is unavailable, and, therefore, for present practical purposes, represents a loss which has to be reckoned with. It may be possible, and quite probable, that in other mining regions this

loss may not occur, but I put some emphasis on the point, as investigators in other countries will do well to closely study the nature of the ore they are dealing with before erecting their plant, and thereby probably save unnecessary expense on that head, to say nothing of costly experiments.

To illustrate the question of extraction by an example, I will suppose that we are dealing with a Rand mine whose ores contain an average assay value of 16 dwts. per ton.

I am safe in saying that the average extraction in the battery by plate amalgamation ranges from 60 to 70 per cent., especially from the pyritic ores; and, assuming that the battery recovers 10 dwts. per ton, or about 63 per cent., this would leave 6 dwts. in the tailings. Assuming now that 1.000 tons tailings at 6 dwts. go to the cyanide works, containing 6,000 dwts., 30 per cent. of these, or 300 tons at 6 dwts., containing 1,800 dwts., go to the slime-pit, leaving 700 tons of tailings at 6 dwts., containing 4,200 dwts., which go to the leaching-vats. Of these 4,200 dwts., 70 per cent. is recovered by cyanide. equal to 2,940 dwts., so that the amount recovered from the original 1,000 tons of tailings represents 50 per cent. in round figures, or 3 dwts, out of the 6 dwts. Add to this the 10 dwts. won on the plates, and we have 13 dwts., or a total recovery of 82 per cent, of the gold contained in the ore, which, when the slime treatment shall have been solved will be raised to about 90 per cent., which percentage can be considered highly satisfactory.

Exact figures as to the actual extraction of the gold from the various establishments are not easily obtainable, but from the information I have gathered, I should consider that 85 per cent. would be the average.

Cost of Treatment.—This necessarily varies at every plant, and mainly depends (1) on the size of the plant; (2) on the facilities for handling the tailings in charging and discharging; while (3) the principal item is the consumption of cyanide.

The consumption of cyanide, I should judge, varies from

<sup>3</sup>/<sub>4</sub> to 1<sup>1</sup>/<sub>4</sub> lbs. per ton of tailings treated. The commercial article, as sold in Johannesburg, costs 2s. per lb.

The consumption of zinc is about 0.4 to ½ lb. per ton of ore, and costs 4½d. per lb. in Johannesburg. At the Kleinfontein works the consumption of cyanide is only ½ lb. per ton. The consumption depends greatly on the washing, and the alkaline wash destroying organic matter. The cost, in large works treating above 10,000 tons and over, would be 4s. per ton; in works treating 5 to 7,000 tons, 5s. to 6s.; in works treating 3 to 4,000 tons, 6s. to 7s.; while in smaller establishments the cost may even be higher.

The cost of zinc precipitation can be estimated at from  $1\frac{1}{2}d$ . to 3d. per ton.

The Robinson Cyanide works treated, in 1893, 55,200 tons of tailings; in which was contained fine gold, 20180.06 oz., and from which was extracted fine gold, 13872.66 oz.; extraction, 68.7 per cent.; bullion returned, 17921.20 oz.

The cost of cyaniding was as follows:-

								To	tal.		Cost per ton.			
Wages (v	vhites	and	nativ	es, in	cludi	ng foo	od)	£ 3,406	s. Il	d. 8	£	s. I	d. 2•81	
General s	tores,	assay	mat	erial,	&c.	•		1,408	9	I	0	0	6.12	
Fuel .		•						1,204	19	9	0	0	5.24	
Cyanide-	-64,41	ıılbs.	equ	al to 1	·16 ll	o. per t	ton	5,563	0	2	0	2	0.19	
Zinc-12,	521 ll	bs. eq	ual t	0 0.2	3 lb.	per to	n.	260	4	3	О	0	1.13	
Contracto	r (filli	ing an	ıd di	scharg	ging v	rats)		4,325	I 2	2	0	I	6∙81	
Royalty	•	•	•	•		•	•	3,768	I	I		I	4.38	
		Tota	1			•	£	19,936	18	2	0	7	2.68	

Cost of Plant.—A plant to treat 3,000 tons monthly would cost £5,000; a plant to treat 5,000 tons monthly, £7,500; a plant to treat 7,000 tons monthly, £10,000; and a plant to treat 16,000 tons monthly, £18,000. The figures being, of course, in each case approximate only.

### CHAPTER III.

## THE SIEMENS-HALSKE PROCESS.

The Electric Precipitation of Gold from Cyanide Solutions.—This process has been in practical operation at the Worcester mine for several months, where I have had the opportunity of observing and testing the operations. At the Meyer and Charlton mine, May Con, and at the Metropolitan, it will very shortly supersede the zinc precipitation method.

Upon reference to the plans (Plates V. and VI.—see pages 42 and 44) it will be noticed that the works are similar to those in use for the MacArthur-Forrest process, the only alteration being in the extractor-house.

For much of the information contained in this Chapter, I am indebted to a lecture by Mr. A. Von Gernet, delivered a few months ago before the Chemical and Metallurgical Society of South Africa. Great credit is to be given to Mr. Von Gernet for the ability which he has displayed in developing this new process on the gold fields.

How the Process was Discovered.—In 1887 Dr. Siemens, the eminent electrician of Berlin, found that the gold anodes, used in electro-plating at his works, lost weight when standing in the cyanide liquor without any electric current passing through the bath. This fact of course corroborated the statements made by chemical authorities that gold is soluble in cyanides, and induced him to try the use of cyanide solutions for the extraction of gold from ores.

He found that the zinc method, as introduced by MacArthur-Forrest, only gave good results with strong solutions, while the

electric precipitation was equally effective with either strong or weak solutions, and its efficiency was not lessened by the presence of caustic soda.

He subsequently introduced the process in Europe, Asia, and America, and during the year 1894 erected a plant capable of dealing with 3,000 tons of tailings per month at the Worcester mine, near Johannesburg, Transvaal.

The Action of the Electric Current on the Gold Solution.—The electric current decomposes a solution of a metallic salt, the metal being deposited on the negative pole, while the metalloid is liberated at the positive pole of the electrolytic cell. In a fixed time, a given electric current will deposit a certain quantity of metal, which quantity varies for different metals in direct proportion to their electro-chemical equivalents. This law holds good only for solutions strong in metal, but with very dilute solutions, as in use in the cyanide process, the current does not find sufficient of the metallic compound present at the electrodes, and consequently decomposition of water also takes place. For this reason, to make the efficiency of the precipitation as great as possible, constant diffusion of the solution is requisite.

In order, therefore, to create an artificial diffusion, a mechanical movement of the solution is important, and the most economical and convenient way of effecting this is to allow a slow but steady flow through the precipitation-boxes. But it is still more important to give a very large surface to the electrodes. In fact, a better effect is obtained by doubling the number of plates than by increasing the current tenfold.

Why Mercury cannot be used as a Cathode.—Mercury cathodes are not practical, for the reason that such an enormous quantity of this costly metal would have to be employed that the recovery of the gold would become difficult.

To precipitate 100 tons of cyanide solution containing 5 dwts. of gold per ton of solution in 24 hours, about 24,000 square feet of mercury surface is required. If the bottoms of

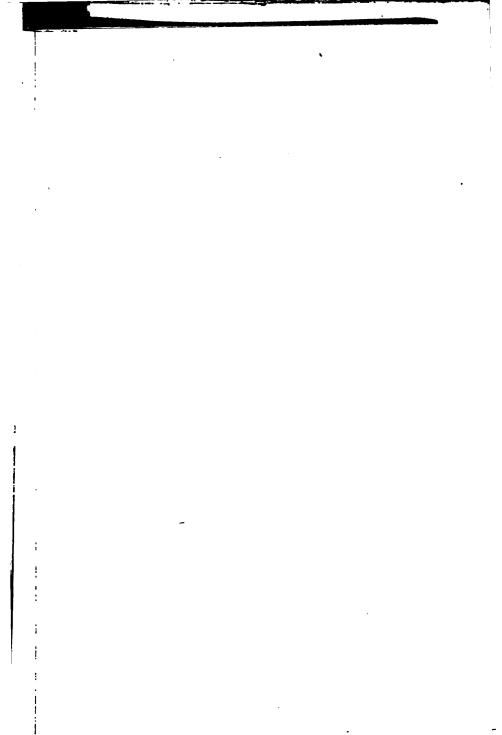
the precipitating-boxes were covered with mercury, it would be necessary to have it at least a quarter of an inch deep, to make up for differences of level, and thus ensure that the whole surface should be covered. This requires over 200 cubic feet of mercury, weighing 80 tons. At the end of a month there would be 750 oz. of gold in this enormous quantity of mercury; and even straining it most carefully, the gold would be so finely diffused that I doubt whether much of the gold would remain in the straining filter; besides, the practicability of such an operation is questionable. I do not consider the initial outlay of the mercury, and the large loss sure to result when handling such large masses of this metal.

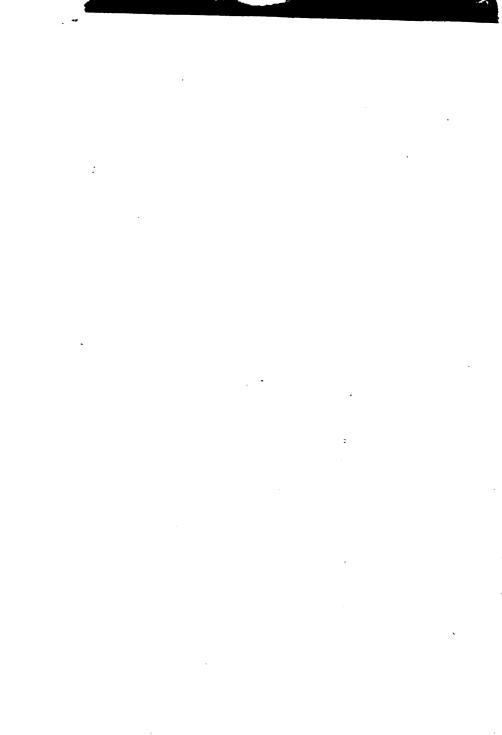
The vertical position in which metallic plates can be placed has the great advantage of keeping the surface of the cathodes clean, as any solid matter entering the boxes in suspension sinks to the bottom of the latter, no obstruction being offered to its downward course. Sheets of solid metal (as copper), coated with mercury, have also been tried, but have been unsuccessful because the mercury, owing to the action of the current, will penetrate the copper and form a dry amalgam which does not adhere to the plates.

# Conditions which the Metal Cathode must Fulfil.—

- 1. The precipitated gold must adhere to it.
- 2. It must be capable of being rolled out into very thin sheets to avoid unnecessary expense.
  - 3. It must be easy to recover the gold from it.
- 4. It must not be more electro-positive than the anode, in order to prevent return currents being generated when the depositing current is stopped.

The most suitable metal for the purpose is lead, rolled out in very thin sheets, and this is accordingly used in the Siemens-Halske process, meeting all the requirements of the case. The lead sheets are fastened in light wooden frames. There are three sheets of lead, 2 by 3 ft. in each frame, giving each frame a surface of 18 ft., and 87 frames, which are in each precipitating-box, will expose a surface of 1,566 square feet.





Each frame holding three lead sheets at 1 lb. weight each, makes 261 lbs. of lead in each box. See Plate VII. (p. 46).

The Anode.—Not less important is the question of anodes. By the action of the current, a metalloid is liberated at the positive electrode, and the latter, when a metal, begins to oxidize. Carbon could be used as an anode, but it will not withstand the action of the current, and soon crumbles into a fine powder, which decomposes cyanide. This finely-divided carbon is in suspension, and cannot be removed from the solution by filtration. Zinc used as an anode forms a white precipitate of ferro-cyanide of zinc by the reaction of zinc oxide upon ferro-cyanide, formed during the leaching. Similarly, iron anodes form Prussian blue by the reaction of oxide of iron and ferro-cyanide. In consequence of this reaction the amount of ferro-cyanide in the cyanide solution does not increase.

From the Prussian blue the cyanide can be recovered by dissolving it in caustic soda, then evaporating the solution, and finally smelting with potassium carbonate.

This last process has been carried out only on a small scale, about 50 lbs. at a time, but a nice clean cyanide of potassium is obtained. In the treatment of tailings this regeneration of cyanide is not of great importance; but with concentrates, which decompose the solution with formation of ferro-cyanide, it will effect a considerable economy.

Electric Current required for Precipitation. — In order to precipitate the gold from cyanide solutions only, a very weak current is required, that is to say, a density of about 0.06 ampère per square foot. With cathodes about 1½ in. apart, 7 volt is sufficient to produce this current strength.

The advantages gained by using such a weak current are:—

- 1. The gold is deposited hard on the plates.
- 2. The iron anodes are preserved for a long time, as their waste is in proportion to the current strength. In a plant

treating 3,000 tons per month, 1,080 lbs. of iron are destroyed in that period.

3. Little power is required. 746 Watts equal 1 horse-power. A 3,000-ton plant requires 2,400 Watts, equal, theoretically, to 3½ horse-power, and actually requiring about 5 indicated horse-power.

Advantages of Electrical Precipitation. — The most important feature of electrical precipitation is, that it operates on the solution quite independently of the amount of cyanide or caustic soda it contains. Precipitation by means of a chemical reaction is invariably more complete with a solution strong in cyanide than with a weak one, but with electricity is absolutely of no importance whatever. Therefore, in the treatment of tailings, very dilute solutions can be used, the only limit being a sufficient amount of cyanide to dissolve the gold satisfactorily. Moreover, however acid the solution may be when entering the boxes, the precipitation takes place equally as well, the same amount of gold being recovered as from a neutral or alkaline solution. There are none of the complications arising from the formation of lime and alumina and hydrate of iron, which, under similar circumstances, sometimes occasion so much trouble in the zinc process.

A solution, containing 0.03 per cent. of cyanide, will dissolve gold just as effectively as a solution containing 3 per cent., provided a longer time is allowed for treatment. In the first case, the decomposition of cyanide in the tailings is much less than in the second, and a corresponding economy is effected. The moisture in the original tailings being usually about the same as that contained in the residues, there is, as a rule, no chance to give a large water wash after the cyanide treatment is finished. The residues discharged contain 10 to 15 per cent. moisture, carrying about 1 to 0.05 per cent. cyanide solution when the zinc process is employed. This is equivalent to \frac{1}{4} lb. of cyanide per ton of tailings. This last may be reduced to \frac{1}{40} lb. by using as weak wash a solution containing only 0.01 per cent. of cyanide, which strength will be perfectly

THAT TAKE

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		•		

suitable for electrical precipitation, though difficult to deal with by the chemical method.

Practical experience has taught us that, if we treat an ore containing copper with a strong, say, 2 to 3 per cent. solution of cyanide, the potassic cyanide may be all decomposed, but if we apply a much weaker solution, say \( \frac{1}{2} \) per cent., although the cyanide will be also decomposed, the same extraction of gold will be effected. So that, though in the former case the treatment may appear commercially impracticable, the proper carrying out of the process, by leaching with weak solutions, will prove to be both effectual and economical.

The presence of copper in the ores also may affect, to some extent, the second operation in the process, i.e., the precipitation of the gold.

Mr. Feldtmann mentions in this connection the result of some experiments made by him on some cupriferous ores at the Transvaal Company's plant near Lydenburg, as illustrating the selective affinity which weak cyanogen compounds possess for gold, that although there was sufficient copper mineral present to decompose a solution of potassic cyanide of over 1 per cent. strength, still good results—i.e., 70 per cent. extraction on 18 dwts. ore—were obtained with  $\frac{1}{2}$  per cent. solutions. In this instance there was reason to believe that the gold was dissolved as auric cyanide instead of as auro potassic cyanide, as usually formed.

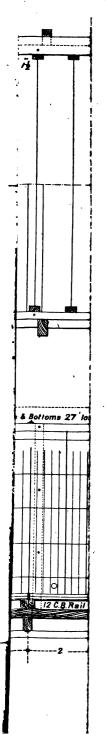
As I have already intimated, it is my belief that the application of the extremely weak solutions, such as are utilised in this process, will lead to important results in the treatment of the so-called rebellious ores. But further investigation of the subject is required.

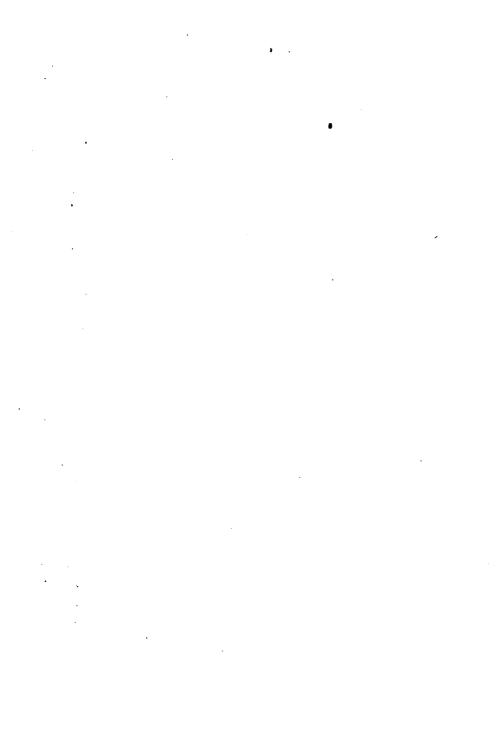
The vats hold 135 tons, and it takes 5 hours to fill them. The tailings first get an alkaline wash of 10 tons, after which wash 70 tons of strong solution of 0.05 to 0.08 per cent. is pumped on; and afterward 20 tons of weak solution, of 0.01 per cent. strength, is pumped on. The total quantity of solution used is 100 tons, and it takes 5½ days to leach, filter, and discharge each vat.

Practical Working Results.—At the Worcester works, which are in charge of Mr. Oswell, there are in use 5 leaching-vats of 20 ft. diameter with 10 ft. staves, each holding 3,100 cubic ft., 135 tons. One tank is discharged and filled every day. The strong solution used contains from 0.05 to 0.08 per cent. cyanide, and the weak washes 0.01 per cent. The actual extraction of fine gold has averaged 70 per cent., while the consumption of cyanide has been ½ lb. per ton of tailings treated.

The precipitation plant consists of four boxes,  $2o \times 8$  ft. wide and 4 ft. deep. Copper wires are fixed along the top of the sides of the boxes, and convey the current from the dynamo to the electrodes. The anodes are iron plates, 7 ft. long, 3 ft. wide,  $\frac{1}{8}$  in. thick. They stand on wooden strips, placed on the bottom of the box, and are kept in vertical position by wooden strips fixed to its sides. In order to effect the circulation of the solution in passing through the box, some of the iron sheets rest right down on the bottom, while others are raised about 1 in. above the level of the solution, thus forming a series of compartments similar to those of a zinc precipitating-box, the difference being that the solution passes alternately up and down through successive compartments.

The iron sheets are covered with canvas to prevent short circuit. The lead sheets are stretched between two iron wires. fixed in a light wooden frame, which is then suspended between the iron plates. The boxes are kept locked, being opened once a month for the purpose of a "clean-up," which is carried out in the following manner:—The frames carrying lead cathodes are taken out one at a time. The lead is removed and replaced by a fresh sheet and the frame returned to the box, the whole operation taking but a few minutes for each frame. By this means the ordinary working is not interrupted at all, and the cleaning out of the boxes, which is necessary in the zinc boxes, is only required at long intervals. The lead, which contains from 2 to 12 per cent, of gold, is then smelted into bars and cupelled. The gold is deposited on the lead sheets as a thin. bright vellow film, which adheres firmly to the lead. The consumption of lead at these works is 750 lbs. per month,





equal to 1\frac{1}{2}d. per ton of tailings, and the working expenses for treating 3,000 tons per month are as follows:—

72711						£			d.	
Filling ar	ia ais	scnar	ging	ieachi	ng-va	ats 125 m	onthly	•	0.10 l	per ton.
Cyanide	•	•			•	75	,,		o·6	,,
Lime	•					15	,,		I · 2	,,
Caustic s	oda					6	,,		0.2	,,
Lead						14	,,		1.1	,,
Iron.						28	,,		2.2	,,
White la	bour					65	,,		5.2	,,
Native w	ages	and f	ood			20	••		1.9	"
Coal.						57			4.6	"
Stores an	d ge	neral	char	ges		41	•		3.2	,,
										••
						£450	or	3	shilling	gs per ton.

At a great many works, where correct sampling and assaying is carried out, the results—that is, the gold won—agree pretty closely with the assay results. At the Worcester the following were the results for the month of August, 1894:—

								Ozs. Dwts. Grs.			
1,350 tons	s tailin	gs take	n fron	ı settling	vats co	ntainir	ng .	443	1	0	gold
1.750	,,	,,	,,	upper ta	ailing d	am cor	taining	498	15	I 2	,,
3,100 ton	s tailin	gs whi	ich ac	cording t	o assay	conta	in .	941	16	12	,,
3,100 ,,	resid	ues	,,	,,	,,	,,	•	239	7	I 2	,,
Balance,											
Actual go	old wo	n, or 7	4.1 pe	er cent.	•			697	15	15	,,

The tailings which were treated assayed from 6 to 8 dwts. The tailings, or residues, after treatment, assayed from 1 to 2 dwts. The solutions which leave the precipitation-boxes still contain some gold, and the analysis in this case showed that the strong solution contained 4 dwts. 8 grs. per ton of solution. The weak solution contained only 10 grs.

On an average, in these works, strong solutions carry from 4 to 5 dwts., and the weak from 0 to 1 dwt.

In taking samples of the residue, it should be borne in mind that the portion of the residues in which the largest proportion of unextracted gold is contained is near the bottom of the filter-vat, say the last 12 to 15 inches.

The best way to sample is to take a sampling-iron and probe every carload as it goes to the works, and to do the like with every car as it leaves the works with the worked residues.

In the subjoined pages (50 to 56) will be found the details of a working scheme for one week, as carried out at the Worcester works; and to follow the method of treatment, I have chosen vat No. 4, as an example.

On the 20th August, 1894, this vat, which holds 135 tons has been filled with tailings. It takes about five hours to fill such a tank. One ton of tailings is equal to 27 cubic feet.

	From 3.15 p.m. till 6.20 p.m. 10 tons of the alkaline wash are pumped into the vat 10 tons									
At	8.10 p.m. s	trong so	olution is	pumped	lon.	:	5 tons			
	3.30 a.m.	,,	,,	,,	٠	•	5 ,,			
		218	r Augu	st, 1894						
At	9 a.m. s	trong so	olution is	pumped	on.		5 tons			
,,	1 p.m.	,,	,,	,,			5 "			
,,	5.20 ,,	,,	,,	,,		•	5 "			
,,	10.15 "	,,	,,	**	•	•	5 ,,			
"	3.40 a.m.	,,	,,	,,	•	•	5 "			
		22N]	Augu	s <b>t,</b> 1894	••					
At	9.15 a.m. s	trong so	lution is	pumped	on.		5 tons			
,,	1.30 p.m.	,,	,,	,,	•		5 "			
,,	6.30 ,,	,,	,,	,,	•	•	5 ,,			
,,	9.35 ,,	,,	,,	,,	•	•	5 ,,			
,,	4 a.m.	"	,,	,,	•	•	5 "			

### 23RD AUGUST, 1894.

At	7.30 a.m. st	rong so	on.	•	5 tons		
		Tot	al strong	solution	•		70 tons
At	1.50 p.m. w	eak sol	ution is p	oumped o	n.		7 tons
,,	7.15 "			7 "			
,,	1.25 a.m.	,,	,,	,,			7 ,,
		To	tal weak	solution	•	•	2I tons
		24TF	AUGU	s <b>t,</b> 1894.			
At	7.30 a.m. w	ater was	sh is pun	ped on			6 tons
,,	1.50 p.m.	,,	,,	,,	•	•	5 "
		Tot	al water	weh			II tone

The tailings are then leached dry and discharged in the morning of the 25th of August.

It takes, therefore, five and a-half days to leach and filter one tank, and the solutions consists of

I.	Alkaline wash					10 tons
2.	Strong solution					70 ,,
3.	Weak solution				•	21 ,,
	Tota	l qua	ntity	of so	lution	— IOI tons

This is by working the Siemens and Halske process.

WORKING SCHEME FOR ONE WEEK AT THE WORCESTER WORKS. From 20th August to 26th August, 1894.

	Volumetric Determination of the Strength of the Solutions.	7 a.m. Tank No. 5. Solution contained oo3 per cent. K. Cy. 11.45 a.m. Tank No. 3. Solution contained oo10 per cent.	1.25 p.m. Tank No. 3. Solution contained o'o24 per cent. K. Cy. 2.45 p.m. Tank No. 3. Solution contained o'o30 per cent.	K. Cy.		
894.	Remarks.					
UST, 1	Wash Water, Tons,	9				
AUG	Weak Solu- tion. Tons.	0 0 0			2	
, 20TI	Strong Solu- tion. Tons.	70	04 01 01 01	15 10 10 5	10	
Monday, 20TH AUGUST, 1894.	Time. Discharging and Filling.	I p.m. 5 tons. 9.5 p.m. 5 ", 3.30 a.m. 5 ", 7 a.m. Water.	1.15 p.m. 5 tons. 9.15 p.m. 5 ", 5.15 a.m. 5 ",	8.50 a.m. 5 tons. 1.25 p.m. 5 tons. 6.10 p.m. 5 ., 9.25 p.m. 5 ., 1 a.m. 5 .,	Discharging and Filling. 1.15 p.m. till 6.20 p.m. from No. 3 solution tank. 8.10 p.m. 5 tons. 10.45 p.m. 5 tons. 3:30 a.m. 5 .,	ıg dry.
	Time. Discharging and Filling.	9.15 a.m. 5 tons. 5.40 p.m. 5 " 12.30 a.m. 5 "	9 a.m. 5 tons. 6 p.m. 5 " 1.30 a.m. 5 "	8.50 a.m. 5 tons. 6.10 p.m. 5 " I a.m. 5 "	Discharging and Filling. 3.15 p.m. till 6.20 p.m. from No. 3 solution tank. 8.10 p.m. 5 tons.   10.45 p.m. 5 tons. 3.30 a.m. 5 .,,	Leaching dry.
	Tank.	No. 1	No. 2	No. 3	No. 4	No. 5

TUESDAY, 21ST AUGUST, 1894.

Volumetric Determination of the Strength of the Solutions.	6.15 a.m. Tank No. 1. Solution	8.30 a.m. Strong solution contained orogo per cent. K. Cy.  10 a.m. Tank No. 4. Solution	2 p.m. Tank No 4. Solution contained orook percent. K. Cy. 2 p.m. Tank No. 4. Solution contained orotz per cent. K. Cy. 7 p.m. Tank No. 4. Solution properties of the contained percent. K. Cy. 7 p.m. Tank No. 6. Solution of the contained percent. K. Cy. 6. Solution of the contained percent.	7-45 p.m. Tank No. 4. Solution contained or 52 per cent. K.Cy. 1.45 p.m. Tank No. 4. Solution contained or 52 per cent. K.Cy. K.Cy. Tank No. 6. Ch. Cy. Tank No. 6. Ch.	y y y p.m. tank No. 4. Sourtion contained o'028 per cent. K. Cy.  Io.30 p.m. Tank No. 4. Solution contained o'03 per cent. K. Cy.
Remarks.					
Wash Water. Tons.		70			
Weak Solu- tion. Tons.		01 01			10
Strong Solu- tion. Tons.		70	<b>\$</b> 000	15 10 10 5	10
Time. Discharging and Filling.	g dry.	1.30 p.m. 5 tons. 9 p.m. 5 " 3.20 a.m. 5 " 7 a.m. Water.	1.15 p.m. 5 tons. 8.50 p.m. 5 3.30 a.m. 5	I p.m. 5 tons.	nd Filling. from No. 3 solu- 10.30 p.m. 5 tons.
Time. Discharging and Filling.	Leaching dry.	9.30 a.m. 5 tons. 5.30 p.m. 5 12.30 a.m. 5 6.10 a.m. to	9.15 a.m. 5 tons. 4.10 p.m. 5 ". 12.40 p.m. 5 ".	9 a.m. 5 tons. 5.20 p.m. 5 3.40 a.m. 5	Discharging and Filling.  3.40 p.m. till 7 p.m. from No. 3 solution tank.  8.40 p.m. 5 tons. 10.30 p.m. 5 tons.  3.50 a.m. 5
Tank.	No. 1	No. 2	No. 3	No. 4	No. 5

1804.
August.
22 ND
WEDNESDAY,

Volumetric Determination of the Strength of the Solutions.	6.15 a.m. Tank No. 2. Solution contained 0.038 per cent. K. Cy. 7.15 a.m. Tank No. 2. Solution contained 0.023 per cent.	K. Cy.	tained o'o54 per cent. K. Cy.  10.15 a.m. Tank No. 5. Solution contained o'o10 per cent.  K. Cy.	130 p.m. 1ank No. 5. Solu- thon contained 0.016 per cent. K. Cy. 3.30 p.m. Tank No. 5. Solu- tron contained 0.026 per cent.	4.30 p.m. Tank No. 5. Solution contained o'030 per cent. K. Cy.
Remarks.					
Wash Water. Tons.			j , , ,	!	
Weak Solu- tion. Tons.	10		01 01 01		
Strong Solu- tion. Tons,	5		0,	01 00 5	10 10 5.
Time. Discharging and Filling.	Discharging and Filling. p.m. till 6 p.m. from No. 3 tank. p.m. 5 tons.   9.45 p.m. 5 tons. a.m. 5 "	g dry.	1.40 p.m. 5 tons. 9.55 p.m. 5 ", 3.30 a.m. 5 ", 7 a.m. Water.	1.30 p.m. 5 tons. 9.35 p.m. 5 ",	1.15 p.m. 5 tons. 9.25 p.m. 5 ",
Time. Discharging and Filling.	Discharging and Filling. 3.50 p.m. till 6 p.m. from No. 3 tank. 8 p.m. 5 tons. 9.45 p.m. 5 tons	Leaching dry.	2 a.m. 5 tons. 1. 6.20 p.m. 5 9. 12 midnight 5 3. 6.5 a.m. to 7 ?	9.15 a.m. 5 tons. 6.30 p.m. 5 " 4 a.m. 5 "	9.30 a.m. 5 tons. 5.20 p.m. 5 ". 2.50 a.m. 5 ".
Tank.	No. 1	No. 2	No. 3	No. 4	No. 5

THURSDAY, 23RD AUGUST, 1894.

Volumetric Determination of the Strength of the Solutions.	6.30 a.m. Tank No. 3. Solution contained 0.044 per cent. K. Cy. 7.45 a.m. Tank No. 3. Solution contained 0.024 per cent. K. Cy.	8.45 a.m. Tank No. 1. Solution contained o'oos per cent. K. Cy. 12 a.m. Tank No. 1. Solution contained o'oos per cent. K. Cy.	4.30 p.m. Tank No. 1. Solution contained o'o'f per cent.	6.15 p.m. Tank No. 1. Solution contained o'024 per cent. K. Cy. Tank No. 1. Solution 7.30 p.m. Tank No. 1. Solution from the contained o'025 per cent.	K. Cy.  8.30 p.m. Strong solution 0.054 per cent. K. Cy.
Remarks.					
Water. Tons.					
Weak Solu- tion. Tons.		10		14	
Strong Solu- tion. Tons.	15 10 10	10		65	\$0.00
Time. Discharging and Filling.	9.30 a.m. 5 tons, 1.25 p.m. 5 tons. 5.30 p.m. 5 9.35 p.m. 5	Discharging and Filling. 4 p.m. to 5.45 p.m. 10 tons from No. 3 tank. 6.45 p.m. 5 tons. 9.25 p.m. 5 tons.	ng dry.	7.15 p.m. 7 tons.	I.40 p.m. 5 tons. 9.15 p.m. 5 ", 3.30 a.m. 5 ",
Time. Discharging and Filling.	9.30 a.m. 5 tons. 5.30 p.m. 5 ". 12 midnight 5 ".	Discharging 4 p.m. to 5.45 p.r. No. 3 6.45 p.m. 5 tons. 1 12.10 a.m. 5 "	Leaching dry.	7.30 a.m. 5 tons. 1.50 p.m. 7 "	9.15 a.m. 5 tons. 2.20 p.m. 5 ". 12.20 a.m. 5 ".
Tank.	No. 1	No. 2	No. 3	No. 4	No. 5

FRIDAY, 24TH AUGUST, 1894.

Volumetric Determination of the Strength of the Solutions.	4.30 a.m. Tank No. 4. Solution contained o'odq per cent. K. Cy 6.30 a.m. Tank No. 4. Solution contained o'odq per cent. K. Cy.	7.43 a.m. Tank No. 4. Solution contained o'062 per cent. K. Cy. 11.15 a.m. Tank No. 4. Solution contained o'046 per cent. K. Cy. 11.30 p.m. Tank No. 4. Solution	ap.m. Tank No. 4. Solution contained oo4 per cent. K. Cy. 4.30 p.m. Tank No. 4. Solution contained oo40 per cent. K. Cy.	S.45 p.m. Strong solution contained o'o54 per cent. K. Cy. 8.45 p.m. Strong solution contained o'o54 per cent. K. Cy.	12.5 a.m. 1ank No. 4. 0'030 per cent. K. Cy. 12 p.m. Tank No. 2. Solution contained 0'026 per cent. K. Cy. 2.10 p.m. Tank No. 2. Solution contained 0'035 per cent. K. Cy.
Remarks.					
Wash Water. Tons.				5	z,
Weak Solu- tion. Tons.	!		01	21	01 01 01
Strong Solu- tion. Tons.	<b>6</b> 000	15 10 10 5	5	70	67
Time. Discharging and Filling.	1.15 p.m. 5 tons. 9.20 p.m. 5 ", 3.30 a.m. 5 ",	1.30 p.m. 5 tons. 9.30 p.m. 5 "	arging and Filling. 5.50 p.m. from No. 3 tank. 5 tons. 9.45 p.m. 5 tons. 5	m. 6 tons. Water. 5 ,, ,,	1.45 p.m. 5 tons. 10 p.m. 5 ", 4.25 a.m. 5 ", 7.45 a.m. Water.
Time. Discharging and Filling.	9.40 a.m. 5 tons. 5.20 p.m. 5 ". 12.30 a.m. 5 ".	9.55 a.m. 5 tons. 5.30 p.m. 5 ". 5.45 a.m. 5 ".	Discharging and Filling. 3.15 p.m. to 5.50 p.m. from No. 3 tank. 7.20 p.m. 5 tons. 9.45 p.m. 5 tons.	7.30 a.m. to 8.30 a.m. 6 tons. Water. 1.50 p.m. Leaching dry. "	10.10 a.m. 5 tons. 5.40 p.m. 5 " I a.m. 5 " 6 a.m. to
Tank.	No. 1	No. 2	No. 3	No. 4	No. 5

SATURDAY, 25TH AUGUST, 1894.

Volumetric Determination of the Strength of the Solutions.	6.45 a.m. Tank No. 5. Solution contained 0.046 per cent. K. Cy. 7.45 a.m. Tank No. 5. Solution contained 0.042 per cent. K. Cy. 10 a.m. Tank No. 5. Solution contained 0.034 per cent. K. Cy.	9.30 a.m. Tank No. 4. Solution contained 0.026 per cent. K. Cy. 8 a.m. Strong solution con-	8.15 a.m. Tank No. 3. Solution contained 0.008 per cent. K. Cy. 10.45 a.m. Tank No. 3. Solution	12.30 a.m. Tank No. 3. Solution contained or of per cent. K. Cy. 2.30 p.m. Tank No. 3. Solution contained or of per cent. K. Cy.	4.30 p.m. rank IVO. 3. Southon contained 0.033 per cent. K. Cy.
Remarks.					
Wash Water. Tons.	0				
Weak Solu- tion. Tons.	222			01	
Strong Solu- tion. Tons.	70	00 00 00 00 00 00 00 00 00 00 00 00 00	15 10 10 5	10	
Time. Discharging and Filling.	1.15 p.m. 5 tons. 8.30 p.m. 5 " 3.10 a.m. 5 " 7.30 a.m. 108.10a.m. 4 tons. Water.	1 p.m. 5 tons. 9.5 p.m. 5 4.30 a.m. 5	\$ tons. 12 noon. 5 tons. 5 9.15 p.m. 5 5	Discharging and Filling. till 8.15. p.m. from No. 3 tank. m. 5 tons   I a.m. 5 tons m. 5 .,	ng dry.
Time. Discharging and Filling.	8.30 a.m. 5 tons. 6 p.m. 5 " 11.30 p.m. 5 " 5 a.m. to 6 a.m. 6 tons.	9 a.m. 5 tons. 1 3.35 p.m. 5 9. 11.45 p.m. 5 4.	8.15 a.m. 5 tons. 5.45 p.m. 5 12.45 a.m. 5	Discharging and Filling. 6 p.m. till 8.15, p.m. from No. 3 tank. 10.5 p.m. 5 tons   1 a.m. 5 tons 4.30 a.m. 5	Leaching dry.
Tank.	No. 1	No. 2	No. 3	No. 4	No. 5

SUNDAY, 26TH AUGUST, 1894.

Volumetric Determination of the Strength of the Solutions.	6.40 a.m. Tank No. 1. Solution	8.15 a.m. Tank No. 1. Solution contained code per cent. K. Cy. 9.50 a.m. Tank No. 1. Solution contained cods per cent. K. Cy.	tained o'cos per cent. K. Cy.  II.15 a.m. Tank No. 4. Solution contained o'cos per cent.	2.30 p.m. Tank No. 4. Solution contained 0016 per cent. K. Cy. 4.15 p.m. Tank No. 4. Solution contained 0028 per cent. K. Cy.	ontained o'034 per cent. K. Cy.	
Remarks.						
Wash Water. Tons.		9				
Weak Solu- tion. Tons.		2 2 2			0 4 9 0 4 9	
Strong Solu- tion. Tons.		70	<b>\$</b> 555	15 10 10 5		
harging ing.		5 tons. 5 ", 5 ",	5 tons.	5 tons. 5 "	3 tank. 3 tons.	
Time. Discharging and Filling.	Leaching dry.	1.15 p.m. 9.30 p.m. 3.45 a.m.	1 p.m. 9.20 p.m. 3.30 a.m.	12.45 p.m. 9.10 p.m.	Discharging and Filling. 5.30 p.m. till 8.20 p.m. from No. 3 tank. 9 p.m. 3 tons. 10 p.m. 3 tons. 11 p.m. 4 " 2.10 p.m. 3 " 3.15 p.m. 3 "	
larging ng.		5 tons. 5 " 5 "	5 tons. 5 "	5 tons. 5 ", 5 ",	Discharging .m. till 8.20 p. p.m. 3 tons. p.m. 4 p.m. 3	
Time. Discharging and Filling.		9.40 a.m. 5 tons 5.30 p.m. 5 " 12.50 a.m. 5 " 6 a.m. till 7 a.m.	9.25 a.m. 5.5 p.m. 12.35 a.m.	9.10 a.m. 5.15 p.m. 12.20 p.m.	Disc 5.30 p.m. till 9 p.m. 11 p.m. 2.10 p.m.	
Tank.	No. 1	No. 2	No. 3	No. 4	No. 5	

## CHAPTER IV.

#### PARTICULARS OF OPERATIONS AT VARIOUS WORKS.

Crown Reef Works.—At these works\* the tailings undergo the following treatment by the direct process.

The 120 stamp battery crushes on an average 17,000 tons of ore monthly. Of this quantity, 12,000 tons of tailings go to the cyanide works, which assay from 4 to 4½ dwts. The slimes, amounting to 5,000 tons, assaying 3½ dwts., go to the reservoirs.

The tailings from the battery pass through 3 spitzlutten, one placed in front of the other, and here a classification of the tailings is effected. From the spitzlutten the stream is divided and passed to the separators or spitzkasten, of which there are four, placed in front of the leaching-tanks. At the bottom is a T-piece, with a nozzle screwed on either side, and a rubber hose, the discharge of which can be regulated by a clamp, so that the stream can be diverted into one or the other of the tanks. Mr. Williams claims that he eliminates more slimes by this system of direct filling, and that the tailings which he retains are richer. He claims that 70 tons of his tailings contain more gold than 75 tons of tailings saved by the intermediary tanks, and that is the reason why he discarded them. According to these figures he saves the treatment of 855 tons of tailings every month.

From the spitzkasten the tailings discharge into six large cement filter-tanks, each 40 ft. from front to back, 34 ft. wide, and 10 ft. high, measured inside the tank. At the bottom the

\* For the information in regard to these works I am indebted to Mr. Williams, the chemist in charge.

tanks are narrower, as the inside is built of sloping walls. The side walls are 4 ft. thick at the bottom and 18 in. at the top. Their capacity when full is about 500 tons of tailings.

The sides and walls are of brickwork set in hydraulic lime, and plastered with cement plaster. The bottom is of cement concrete at least 12 in. thick. The concrete is made up of three parts of treated tailings sifted, five parts broken stone down to 2 in. cube, and one part of Portland cement. It is mixed, put in place, rammed, and levelled up with a grout mixed two of sand and one of cement. The bottom of the cement-tank slopes to a central gutter, having a 2-in, fall from the front and back to the centre, where a gutter, 4½ in. wide, starts at the division walls of the tank with a fall of 4 in. towards the centre, where is a small pit 15 × 15 in. and 12 in. deep, formed in the cement bottom, connected with a 4-in. pipe bedded in the concrete, and through which the 2½-in. solution pipe is threaded.

The filter bottom averages 5 to 6 in. in thickness, and is made by filling in with broken stone a 3 by 3 in. wooden combing, laid in cement mortar and bolted to the bottom of the tank all round. Over this is laid the filter-cloth.

Each tank is provided with three iron gates, through which the residues are trammed to the dump. Each gate is suspended by a vertical pulley from a horizontal axle running on wheels, which roll to and fro on rails running on the top of the wall on either side of the door.

A cast-iron frame is built into the brickwork round the doorway, and the door made water-tight by wedging against rubber packing. Inside the filter-tanks are laid three 18-in. tramway tracks on longitudinal sleepers, which remain in the filter, and are only removed when the filter-cloth is to be raised. In the doorway is fitted a short piece of rail composed of 2 in. square iron, provided with pins projecting from its lower surface, which pins are dropped into holes in iron plates fitted on the sills inside and outside the doorway. This piece of rail is removed when it is necessary to open the door. To still further reduce the quantity of slimes treated, there are

slime gates in the tanks. These are between the doors of the tanks, and are simply a series of wooden slats about 4 in. wide, sliding in a vertical groove  $2\frac{3}{4}$  in. from the wall of the tank. During the filling of the tank with tailings, any slimes which remain in suspension flow over into this sluice-gate, which is raised by additional slats as the tanks fill higher and higher, and pass away by a 4-in. drain-pipe to the reservoir.

Even distribution of the tailings of the tanks is effected by constantly moving the nozzle of the charging hose.

Any leakage of cyanide from the doors of the tanks is provided for by a gutter 6 in. wide being formed in the concrete, outside the wall, and about 12 in. from it. This, starting at one end of the row of tanks, falls about 12 in. in its total length, and terminates in a small brick water-tight tank, 2 ft. square by 2 ft. 6 in. deep, from which a pipe leads to the extractor-house.

The solution drainage pipes are 2½ in. in diameter, and, until outside the tanks, are threaded through a 4-in. pipe.

The tailings are under treatment for 102 hours. Each tank of 500 tons gets about 400 tons of solutions and washes.

The strong solution contains 0.3 per cent. K Cy The weaker solution contains 0.15 per cent. K Cy The weakest solution contains 0.05 per cent. K Cy

The final wash water amounts to 25 tons.

The treatment in the leaching-vats is as follows:—It takes 30 hours to fill one of these vats. No. 3 vat, for instance, holds 550 tons of tailings. Say at 12.30 on the 29th August, 50 tons of 0.05 per cent. of cyanide solution is run on to drive out the water. When the solution is run through 75 tons of strong solution, 0.3 per cent. is put on. On the 30th August, 75 more tons of 0.3 per cent. is put on. On the 31st August, 65 tons of 0.15 per cent. is put on. From the 1st to the 3rd September, 300 tons of 0.05 per cent. solution, inclusive of 25 tons of wash water, are put on. Altogether the treatment takes 6 days, the total quantity of solution employed being 565 tons for this particular tank, in which the tailings assayed 54 dwts.

and the residues 1.1 dwts. An extraction of 80 per cent. was obtained, which is rather above the average.

The quantity of tailings treated in each tank is tallied by counting the number of trucks which are discharged. From the leaching-vats the solution runs to the extractor-house, where there are three large stock solution-tanks excavated in the ground, brick-lined and cemented, and of the same dimensions as the lixiviation tanks.

Into No. 1 tank are run all solutions up to 0.08 per cent. K. Cy., which makes up an average of 0.05 per cent. with the weaker solutions.

No. 2 tank receives all solutions from 0.08 to 0.2 per cent. K. Cy., making up an average of 0.15 per cent. K. Cy.

No. 3 tank receives everything from 0.2 per cent. upward, and is made up to 0.3 per cent. by adding extra strong solution from the dissolving tank, and the same is agitated by means of a circulating pump.

There is also a vacuum well, consisting of two cylindrical iron receivers connected to an air-pump, in order to create a vacuum in case the filter is stopped up, or when it is necessary to assist percolation, or when it is necessary to dry the ore before discharging.

From the stock solution-tanks centrifugal pumps take the solution either from one or the other, and pump it into the filter-tanks. Before the solution passes into the extractor-boxes, the same flows into three small wooden tanks, where any impurities settle.

At this mine 50 to 60 per cent. of the gold leaving the battery is obtained in the cyanide works. The consumption of cyanide is claimed to be ½ lb. per ton treated and 4½ oz. zinc per ounce of gold won. The works produced 2,914 oz. gold in August, 1894. The works are nicely planned and splendidly constructed, and the mechanical details have been, no doubt, under the supervision of a skilled man, and may be considered as one of the fine plants on the gold-fields, but they gave me the impression that their cost of erection must be far in excess of those plants which use wooden tanks.

Simmer and Jack Works.—When these works were erected the Company had a large accumulation of tailings, the results of five or six years' working, crushing first with 50 and afterwards with 100 stamps.

A contract was entered for the treatment of 100,000 tons with the Rand Central Ore Reduction Company, and under the direction of Mr. Butters, one of the finest and best plants on the gold fields was erected within three months. While in Johannesburg, I made this plant the special object of my study, and I am indebted to Messrs. Torrente and Smart, the chemists in charge, for some of the working details here given, as well as for the illustrations of the works given in Plates VIII., IX., X.

The plant has a capacity to treat 600 tons tailings daily, and consists of five vats of 600 tons capacity each; three stock solution storage-tanks of 300 tons capacity each; building containing four precipitation-boxes; three 2-in. centrifugal pumps, for returning the solutions from the precipitation-boxes to the storage-tanks; one 4-in. centrifugal pump, for pumping the solutions from the storage-vats to the leaching-vats; two lathes for cutting the zinc shavings; engine to work lathes and pumps and a 60 horse-power boiler to supply steam to engine, and two hauling gears.

There is stabling, also, for 40 mules; besides a large compound to house 200 Kaffirs, a manager's house, and 6 rooms for the white employees.

The tailing vats are 42 ft. in diameter and 14 ft. high, firmly bound with iron hoops 1 in. thick. It takes 10 to 11 hours to fill each vat.

The zinc boxes, four in number, are placed in a building which also encloses the stores, office, engine, lathes, and boiler. They are 24 ft. long, 3 ft. 6 in. broad, 2 ft. 6 in. deep. They are divided in 13 compartments. In every compartment there is a tray with a sieve of 20 to 25 holes to the inch at the bottom, on which the zinc shavings are loosely placed till they reach within 6 in. of the top of the boxes. The trays reach within 4 in. of the bottom of the boxes, this space being

left to collect the slimes resulting from the precipitation of the gold on the zinc. The compartments are divided from each other by two boards, the one reaching to the bottom of the box, and the other within  $3\frac{1}{2}$  in. This ensures the greatest amount of contact of the solution with zinc, and therefore an almost perfect precipitation.

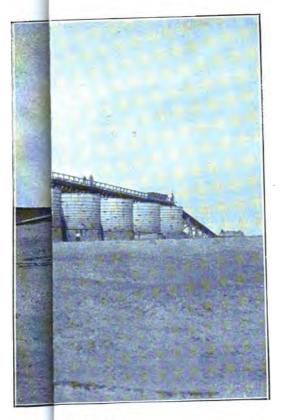
A 4-in. centrifugal pump is used to run the solutions from the storages into the tailing vats, and three small 2 in. centrifugals, return the solutions from the zinc boxes into their respective storages. A donkey pump is used for cleaning the zinc boxes, and three small tanks, called the clean-up tanks, which are used to settle the zinc slimes.

Working of the Plant.—I shall here restrict myself to the actual manipulation of the solutions. The process as carried out at the Simmer and Jack works has to deal (1) with old tailings containing acid; (2) with the tailings coming daily down from the mill, where they keep a certain number of stamps working on free-milling ore (oxidized), and the remainder on pyritic material (blue rock).

The quicklime is ground in a ball mill, constructed by the Gruson works, in Magdeburg, and in every truck of tailings a certain quantity is put, in proportion to the amount of the free acid therein contained.

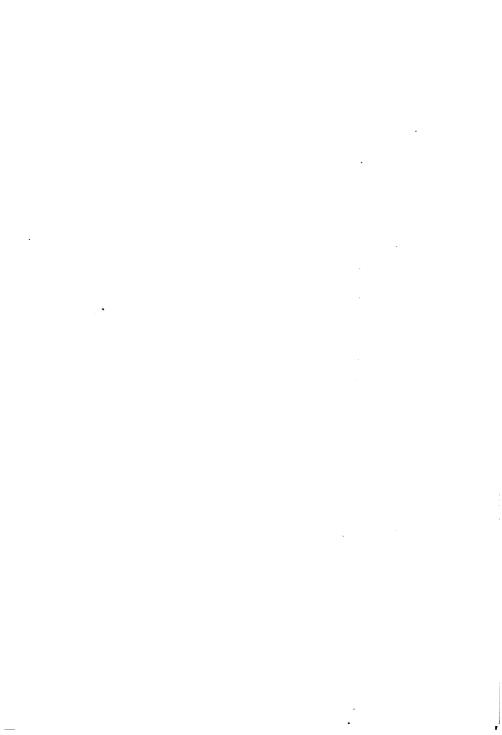
A good practical way of ascertaining this is to take a given quantity of tailings, say I kilo., and place them in an enamelled bucket or basin, adding two or three times their weight in water. Leave them there for two or three hours, occasionally stirring the same, and the blue litmus paper will show if any acid is present. Given the presence of acid: take a weighed quantity of quicklime, finely ground (say 50 grms.), and with a spoon throw a small quantity in the vessel containing the tailings, stirring the same all the time, and keep on adding lime, little by little, until the red litmus paper turns slightly blue, when the acid will be neutralized. By weighing again the remaining lime, the quantity used will be known, and if, say 2 grms. has been used, we shall have that—

# PLATE VIII.

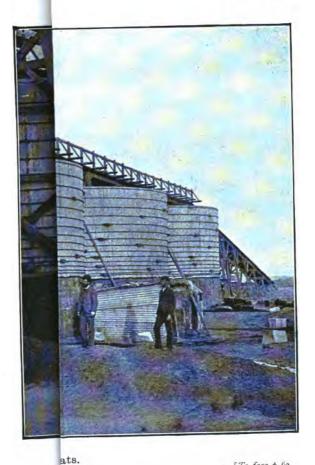


anide Plant.

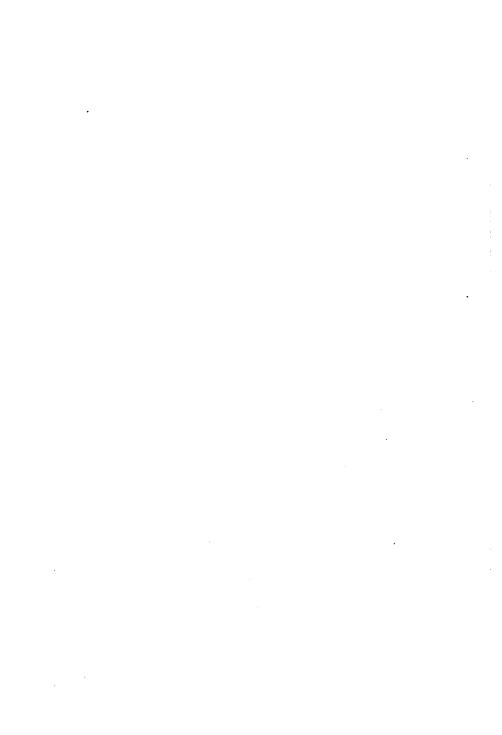
[To face p. 62.



# PLATE IX.



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If I Kg. takes 2 grms. of lime
1,000 Kgs. (I ton), will require 1,000 times more, or
2,000 grams, which is equal to 2 Kgs.

Now, if every truck carries, say, \( \frac{1}{2} \) ton of tailings, it will require 1 kilo. of lime, or 2.12 English lbs. The advantage of supplying every truck with its proper quantity of lime, is to ensure its even distribution through the mass in the vat.

When the vat is full, it is carefully levelled, and then is ready to be treated.

In the case when lime has been used, the tank is filled with water and allowed to stand full for one hour. By this time the acid has been neutralized, and it can be started leaching. The water from the leaching carries, as a rule, a small excess of alkali, and it is pumped back into one of the storages, to be used again in the next tank. To this water the name of alkaline solution, or caustic wash, is given; and, as a rule, the moisture in the tailings is enough to keep the amount in hand constant, so that very seldom is it necessary to add any more water to it. It is necessary to keep the solution in a storage tank, so as to avoid any loss of gold, as it always carries with it whatever gold may have been left in the filters, &c., from previous treatment. It will also be found that, after treating a few tanks, the solution will contain a small percentage of cyanide.

When the caustic wash is partly drained off, the top of the tailings are turned over with a shovel, so as to break the slimy film which forms; and then follows the solution, whose quantity must be so regulated as to give about \frac{3}{4} lb. of cyanide (or less) per ton of tailings, and be of a strength of '25 to '3 per cent.

This solution is followed by another, which is called weak solution, and whose strength varies between '08 and '15 per cent. of cyanide, and is kept on until the tank is ready to leach dry.

Should the solution in the storages get short, a water wash is given to terminate the operation: otherwise the weak solution is used to the last.

The system followed in pumping up the solutions is to keep the tanks always well covered up from the moment the strong solution is first pumped up. As soon as the tailings show, more solution is added until the full amount has been pumped up. This is followed at once with a weak solution, which, in its turn, must keep the tailings always well covered till the tank is ready to water wash or leach dry.

The time employed in these diverse operations is about as follows:—

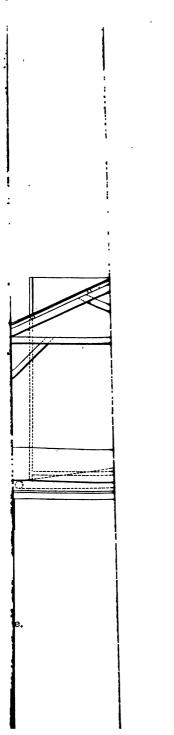
ıst.	Caustic I	Wash	wa	ter or	r othe	rwise	).					
	Pumping		•									3 hours
	Contact											1 hour
	Leaching		•							•		8 hours
2nd.	Strong .	Solut	ion.									
	Is pump			t sto	pping	the	leachi	ing u	ntil a	ll is	on,	
	takes a	bout										8 hours
			(4	Amor	ınt us	ed ab	out I	60 to	ıs.)			
3rd.	Weak So	lutio	n.									
•	Is kept u			ng as	prev	iously	expla	ained	abou	t.		40 hours
4th.	Water H	Vash	(if n	ecesso	iry).							
•	Leach dr											24 hours
	Time it t	ook t	o loa	d tan	k.							12 ,,
	Total from	m the	tim	e of	starti	ng fill	ing ti	ill rea	dy to	disc	harge	96 hours
T)	otal amo	ınt o	f stro	ng ar	ıd we	ak so	lution	betw	een 5	00 a	nd 60	o tons.)

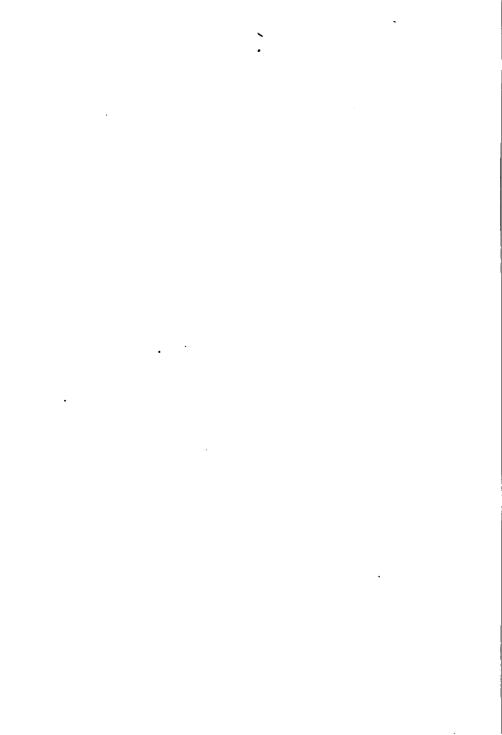
Precipitation of the Gold from the Solution.—From the bottom of the tailing tanks, and under the filter, a pipe 2 in. in diameter carries the solution to the zinc boxes, four in number, as already mentioned.

The object of these four zinc boxes is to return the solutions to their respective storages, for which purpose they are connected:—

```
2 with the strong solution storage,
I with the weak solution storage,
I with the caustic wash storage.
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A simple assay will show to the man in charge when to lead





the leachings through the one or through the other box. The boxes have a fall of 9 in. in their full length.

Recovery of the Gold from the Gold Slimes.—The gold slimes are settled in the slime-tank by means of alum or sodium sulphide. After 12 hours the clear portion is syphoned down, and these slimes are then ready for further treatment. The test of the sodium sulphide is lead paper, which takes a brownish colour when the residues in the tank have been saturated.

To make sodium sulphide:-

6½ gals. water,
62½ lbs. caustic soda,
41 lbs. sulphur (broken small).

Boil the water with the soda till dissolved, and then add the sulphur slowly, as it is apt to boil over. Dilute to 40 gals. Smaller quantities in proportion.

The slimes are transferred to a special calcining furnace, where they are dried and the zinc slightly oxidized. They are left to roast at a dull red heat for 4 hours, although this time depends upon the quantity and nature of the slimes. After roasting, they are put into iron dishes and mixed with the smelting fluxes.

To 6 shovels of slimes add-

shovel flour spar,
shovels borax,
shovel carbonate of soda.

If the slimes are hard, and cake, they must be ground fine so as to mix them well with the fluxes.

If the fire gets too fierce while smelting the slimes, and the pots overflow, keep the cover off, and throw a little borax on the top. This overflowing is objectionable, as the slag cakes at the bottom of the grate and chokes the fires. When all the

gold belonging to one lot of slimes has been secured, it is then melted down into one bar.

Langlaagte Estate.—At these works, the pulp on leaving the mill passes into three settling-dams in rotation. The lower retaining walls are composed of sand-bags, and two doors are provided in each dam, one for the passage of the full, and the other for the empty trucks.

The discharge of the slimes, when filling the dams, is regulated by strips of wood in the passages, the same system being employed at the Randfontein Estate. Lines of rails are placed inside these dams; and at right-angles to these, immediately inside the lower walls, runs a line of rails sunk in the ground, on which is placed a travelling-carriage, on to which the tipping trucks are run and brought outside the dams, to be picked up by a mechanical haulage composed of an endless wire rope, and carried to the cyanide works.

Here are ten vats sunk in the ground, five on each side, and when the tailings are treated they are emptied by means of a travelling crane, which lowers the body of 25 cubic feet capacity trucks into the vats, to be filled by Kaffirs, and are then lifted out again and placed on their respective carriage, and conveyed by mechanical haulage to the waste dumps. A vat can be emptied in six hours.

The mechanical haulage is operated by a 25 n.h.p. compound engine, with 24-in. stroke and 25 h.p. boiler, consuming about 13/4 tons coal per day.

Rand Central Ore Reduction Company.—A prominent position in the treatment of tailings and concentrates is occupied by the above Company, which was organised by Mr. Ad. Goerz, Mining Engineer, representing a powerful German syndicate. The Company, who own the Siemens and Halske patents for South Africa, have bought over 1,000,000 tons of tailings, of an average value of 4½ dwts., and 6,173 tons of concentrates, of an average assay and contents of about 4 oz. The technical work is directed by Mr. Charles Butters.

The establishments of the Company are six in number, and include (1) their central works (shown in Plate XI.)—consisting of a chlorination plant and cyanide works, with workshops and foundry, where everything is made for vats, implements, etc., required for the construction of cyanide plants; (2) works at Maraisburg, which were built to treat tailings from the Main Reef, Aurora, and Aurora West Companies; (3) the Simmer and Jack works, already described; (4) the Worcester works, already described; (5) the No. 1 works, which treat tailings from the Pioneer Company; and (6) the No. 2 works, which treat tailings from the United Langlaagte.

During the year ending June 30th, 1894, the Company treated at their six works 311,561 tons of tailings, yielding 52,434.46 oz., of a value of £209,737 16s. 8d.

Dry Crushing.—As proper percolation of the solution can only be obtained by having the pulp coarse enough to allow of easy leaching, several establishments in the United States have adopted dry crushing.

Mr. Philip Argall, M.R.I.A. (see *Mining Journal*, London, October 13th, 1894), says: "The best results hitherto obtained on rather friable oxidized siliceous ores have been produced by the following series of machines:—

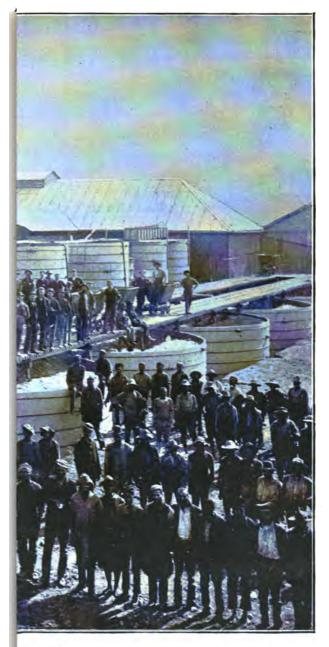
- "(a) An ordinary Blake crusher, which reduces the ore to about  $1\frac{1}{2}$  in.;
- "(b) A three-jaw multiple Blake crusher, which brings the ore down to  $\frac{3}{4}$  in.; followed by
- "(c) A five-jaw multiple, that reduces it to about 6 mesh; and
  - "(d) Rolls that finish the pulp to pass a 30 mesh screen.
- "The ore is screened after each crushing, so that the portion reduced to the desired fineness passes direct to the finished orebin, instead of, as is too often the case, going to the next machine to be reduced to a further and unnecessary degree of fineness. This plant, crushing to 30 mesh, produces only from 5 to 10 per cent. fine enough to pass 200 mesh.

"With soft, clayey ore, from 5 to 15 per cent. of the

finest dust has to be removed before a leachable product is obtained."

The preparation of the ore, in dry as well as in wet crushing, is a very important step in the direct treatment by the cyanide process, and the latter has only been brought to its present state of perfection within the last six months. Owing to the scarcity of water in West Australia, I believe that many mines will have to adopt dry crushing plants, and that the system will reach there a high degree of perfection.

PLATE XI.



ompany.

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#### CHAPTER V.

#### THE CHEMISTRY OF THE CYANIDE PROCESS.\*

Solution of the Gold.—The solubility of gold in a solution of cyanide of potassium has long been known. The presence of oxygen is necessary for the reaction, after the formula:

$$2Au + 4KCy + O + H_2O = 2KAuCy_2 + 2KHO$$
.

That is, a double cyanide of gold and potassium is formed. This was proved by the formation, upon evaporating the solution, of octahedral crystals answering to the formula. From the solution the gold is precipitated by filiform zinc. Based on these two reactions, the MacArthur-Forrest Company have obtained the following patent:—

"The invention consists in subjecting the auriferous or argentiferous ores to the action of a solution containing a small quantity of cyanide, as hereinafter set forth, without any other chemically active agent, such quantity of cyanide being reckoned according to its cyanogen, and the cyanogen being proportioned to the quantity of gold or silver, estimated by assay or otherwise to be in the ores under treatment. By treating the ores with the dilute and simple solution of a cyanide, the gold or silver is, or the gold and silver are, obtained in solution, while any base metals in the ores are left undissolved, except to a practically inappreciable extent;

\* The chemistry of the process has been described by Messrs. Butters and Clennell in the *Engineering and Mining Journal* of October 22nd and 29th, 1892, and from their articles (with their permission) this matter is extracted.

whereas when a cyanide is used in combination with an electric current, or in conjunction with another active chemical agent—such as carbonate of ammonium, or chloride of sodium, or phosphoric acid—or when the solution contains too much cyanide, not only is there a greater expenditure of chemicals in the first instance, but the base metals are dissolved to a large extent along with the gold or silver, and for their subsequent separation involve extra expense, which is saved by our process.

"In carrying out our invention practically, we take the ore in a powdered state and mix with it the solution of cyanide in a vessel made of, or lined with, any material not appreciably acted on by the solution. . . . We regulate the quantity of cyanide so that its cyanogen will be in proportion to the quantity of gold or silver in the charge of ore; but in all cases we dissolve it in sufficient water to keep the solution extremely dilute, because it is when the solution is dilute that it has a selective action, such as to dissolve the gold or silver in preference to the baser metals.

"In dealing with ores containing 20 oz. or less of gold or silver, or gold and silver, per ton, we find it most advantageous to use a quantity of cyanide, the cyanogen of which is equal in weight to from 1 to 4 parts for every 1,000 parts of the ore, and we dissolve the cyanide in a quantity of water of about half the weight of the ore. In the case of richer ores, while increasing the quantity of cyanide to suit the greater quantity of gold or silver, we also increase the quantity of water so as to keep the solution dilute. In other words, the cyanide solution should contain from 2 to 8 parts, by weight, of cyanogen to 1,000 parts of water, and the quantity of the solution used should be determined by the richness of the ore. After the solution has been decanted or separated from the undissolved residues, the gold and silver may be obtained from it in any convenient way, such as evaporating the solution to dryness and fusing the resulting saline residue, or by treating the solution with sodium amalgam.

"Having fully described our invention, what we desire to

claim and secure by letters patent is: The process of separating precious metals from ore containing base metal, which process consists in subjecting the powdered ore to the action of a cyanide solution containing cyanogen in the proportion not exceeding 8 parts of cyanogen to 1,000 parts of water."

Some time after this patent was issued, patents covering the use of zinc, preferably filiform, or threadlike, for a precipitating agent, and the use of caustic alkalies for neutralizing ores containing acids, or acid salts, were granted to Messrs. MacArthur and Forrest. It will be seen, therefore, that their patents cover substantially three points: namely, the use of dilute solutions of cyanide (not more than 8 parts of cyanogen to 1,000 parts of water); the use of zinc, preferably filiform, as a precipitant; and the employment of caustic alkalies for neutralizing acid ores.

Mr. Louis Janin claims,\* that the necessity of oxygen in the reaction which takes place in the solution of the gold in the cyanide has not been proved, and establishes the following formula:—

$$Au + 2KCy + H_2O = KAuCy_2 + KHO + H.$$

More recent investigation has proved that oxygenation of the gold in the process is essential to its success, and the idea has been suggested that an artificial oxygenation of the gold in rebellious ores may lead to the solution of the problem of treating them successfully by the cyanide process.

Mr. R. Feldtmann, in his "Notes on Gold Extraction," gives the following additional formulæ, throwing light on the reac tions which take place in the solutions of gold in cyanide:—

When hydrocyanic acid dissolves gold from acid tailings (assuming that the cyanicide has destroyed all the cyanide of potassium),

 $_{2}Au + _{3}Cy + _{3}C = _{2}AuHCy_{4} + _{3}H_{2}C$ 

an auric hydrocyanide is produced, and this compound is not, or is at any rate only imperfectly, precipitated by zinc. Even rendering the solution alkaline by addition of caustic soda or potash, does not appear in such a case to cause a good precipi-

\* Mineral Industry, 1892. Scientific Publishing Co., New York.

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tation. This may be owing simply to the auric compound being more stable than the aurous salt ordinarily obtained, or may be owing to the absence of free cyanide of potassium.

The addition of alkali to such a solution of auri cyanic acid may be assumed to form auric potassic cyanide in this way:—

$$AuHCy_4 + KOH = AuKCy_4 + H_2O.$$

The addition of an acid to working solutions containing free potassic cyanide, and a certain amount of auro potassic cyanide, would appear—probably indirectly—to convert at least a portion of the latter into auri cyanic acid. Probably the hydrocyanic acid, liberated by the decomposition of the potassic cyanide, combines with the auro potassic cyanide to form auri potassic cyanide:—

$$AuKCy_2 + 2HCy + O = AuKCy_4 + H_2O$$

which, in its turn, acted on by the mineral acid, is converted into auri cyanic acid:—

$$_{2}$$
AuKCy<sub>4</sub> +  $_{2}$ SO<sub>4</sub> =  $_{2}$ AuHCy<sub>4</sub> +  $_{2}$ SO<sub>4</sub>.

Solubility of other Metals and Minerals.—According to Gmelin, zinc, iron, nickel, and copper are dissolved by potassium cyanide, with evolution of hydrogen; cadmium and silver in the presence of oxygen; and tin, mercury, and platinum not at all. Sulphide of silver is dissolved by strong solutions and a sufficient quantity of weak solution. Silver arsenate Ag<sub>3</sub> As O<sub>4</sub>, and silver antimonate AgSbO<sub>3</sub> are readily dissolved by potassium cyanide, as are many of the argentiferous arsenical and antimonial minerals found in nature. Chloride of silver dissolves readily, forming chloride of the alkali and a double cyanide of silver and potassium. While metallic silver, when sufficiently fine, dissolves readily in the solution, that found native in ores is not attacked, unless existing in thin laminæ. The oxides and sulphides of copper are attacked by the solution and dissolved, as is metallic copper.

It is claimed that the presence of copper sulphide in a silver or gold ore prevents the precious metals from going into solution. Although experiments have shown that little or no silver or gold is dissolved in certain ores containing sulphide of copper, this question is by no means settled, as artificially-prepared sulphide of silver is dissolved in actual contact with the copper compounds. Metallic iron is attacked, but very slowly. Ferric hydrate is not attacked by the solution, but ferrous hydrate, formed in the neutralization of the iron salts by alkali, is attacked by cyanide, according to the reaction:—

$$Fe_2HO + 6KCy = K_4FeCy_6 + KOH.$$

Thus ferrocyanide of potassium and caustic potash is formed.

Louis Janin's Experiments on Silver Ores.\*—Mr. Janin made a series of experiments on the applicability of the cyanide process to various types of ores, with the following results:—

Sample No. 1.—Grand Central Mine, Arizona. Siliceous ores, containing considerable quantities of lime and manganese. The silver minerals are principally cerargyrite and argentite, and gave an extraction of 92.6 per cent.

Sample No. 2.—Christy Mine, Silver Reef, Utah. The silver minerals are chlorides, sulphides, metallic silver, in a gangue of sandstone, somewhat discoloured by carbonate of copper. Extraction, 80 per cent.

Sample No. 3.—Horn Silver Ore, Utah. The silver principally as chloride. Extraction, 93 per cent.

Sample No. 4.—Tybo, Nevada. The mineral is principally a sulphide and fahlore. Extraction, 71.8 per cent.

Sample No. 5.—Gangue siliceous with chloride of silver. Extraction, 97 per cent.

Sample No. 6.—Ramshorn, Idaho. Containing galena and carbonate of lead, pyrites, and zinc blende. Extraction, 80 per cent.

Sample No. 7.—Broken Hill, New South Wales. Chloro bromide of silver in kaolin, quartz, and garnet. Extraction, 99.7 per cent.

<sup>\*</sup> From Mineral Industry. The Scientific Publishing Co., New York.

Sample No. 8.—Broken Hill, New South Wales. A siliceous iron ore containing 38 per cent. Fe O. Extraction, 84.6 per cent.

Sample No. 9.—Bullionville, Nevada. Tailings, containing 10 per cent. carbonate of lead, galena, and iron in a siliceous gangue. Extraction, 32 per cent.

Sample No. 10.—Bertrand and Geddes, Nevada. Contains antimoniate of lead, with which the silver was combined. Extraction, 11.8 per cent.

Sample No. 11.—Argenta, Montana. Contains over 40 per cent. lead. Extraction, 5.7 per cent.

Sample Nos. 12 and 13.—Belmont, Nevada. Contains arsenical pyrites, pyrite, blende, and galena, with the silver as fahlore, and arsenical and antimonial ruby forms. Extraction, 35 and 47'5 per cent.

Sample No. 14.—Las Yedras, Mexico. Contains large quantities of carbonate of lime, with the silver in the form of ruby silver, and arsenical pyrites. Extraction, 41.5 per cent.

Sample Nos. 15 and 16.—Ontario and Daly ores, Utah. Contain principally fahlore, more or less decomposed with some lead and zinc. Extraction, 72.5 and 81.1 per cent.

The conclusions and deductions to be derived from a study of the foregoing are, that silver in oxidized surface ores, or where it occurs as a chloride, is readily attacked by cyanide of potassium, and that where no minerals are present which exert an unfavourable influence, this method may prove economical. It must be confessed, however, that even with these conditions it has but a limited range of usefulness. On the other hand, where lead, oxide of copper, or oxides of iron occur, the results are so poor as to preclude the use of the process.

The results obtained from different samples of silver ore from the same mine vary greatly, for a slight increase of an undesirable element, which would not affect amalgamation in the slighest degree, causes a great decrease in the percentage of extraction by cyanide.

In the case of a number of experiments on Daly ores, the

results varied 16.2 per cent., while, with amalgamation, there was a variation of but 2 per cent.

Results of Experiments made with Potassium Cyanide on Gold and Silver Ores.—It would seem probable that, in ores containing both gold and silver, only the oxidized surface ores can be treated with success, both the silver and gold minerals from depth proving refractory. With the majority of these ores the consumption of cyanide would be large, as many minerals other than those of silver are contained in them, and would have a decomposing action upon the solution.

Experiments on Gregory concentrates, consisting of pyrites, some arsenopyrite and chalcopyrite, with traces of galena and blende, gave an extraction of 90 per cent. gold and only 3 per cent. silver.

Delamar, Idaho, consisting of a siliceous and limestone gangue, impregnated with silver sulphides and chlorides, and iron pyrites. Results: gold, 90 per cent.; silver, 83 per cent.

Revenue Mine, Montana—Oxidized surface ores, containing a considerable quantity of iron. Gold, 94 per cent.; silver, 5'2 per cent.

Southern Cross, Montana, contains 40 to 50 per cent. of limonite. Gold, 93 per cent.; silver, 50 per cent.

I am not aware, so far, that the cyanide treatment has been applied on a large scale to the extraction of silver ores.

Treatment of Pyritic Ore Previous to Lixiviation with Cyanide. — Pyritous ore, if but slightly exposed to atmospheric action, always contains free sulphuric acid and soluble salts of iron. To prevent reactions on the solution, it is necessary to leach these ores with water previous to lixiviation with cyanide of potassium, and before washing with an alkali. If the alkali solution were to be added directly to the ore, the consumption of alkali would be extremely large, and the amount of solution necessary, if lime were to be used, would prove inconvenient to handle.

The alkali solution, assuming that caustic soda is used, reacts on basic iron salts, insoluble in water, according to the following reactions:—

$$Fe_2O_3$$
,  $SO_3 + 2NaHO + H_2O = Fe_2(HO)_6 + Na_2SO_4$   
and  
 $Fe_2O_3$ ,  $2SO_3 + 4NaHO + H_2O = Fe_2(HO)_6 + 2Na_2SO_4$ .

Thus ferric hydrate and sodium sulphate (or calcium sulphate, if lime is used) are formed. Sodium sulphate is soluble and passes off with the wash water, but calcium sulphate remains. The hydrate of the sesquioxide of iron is insoluble in water, and, to all appearances, is unattacked by the cyanide solution; but the hydrate of protoxide is dissolved with formation of ferrocyanide of potassium.

Mr. C. W. Merrill precipitated ferrous hydrate by caustic potash from a cyanide solution. The solution contained but a small percentage of free cyanide, however, as it had already acted on ore and zinc in the precipitation of the dissolved gold, and it is unlikely that this reaction, regenerating the cyanide of potassium, which had been rendered inert by the solution of the iron, would occur in a comparatively strong solution.

Treatment of Pyritic Ores by Cyanide of Potassium.

Owing to the rapid decomposition of the pyritic ores when exposed to the atmosphere, precautionary measures have to be taken to treat them effectively. Owing to their successful treatment by the chlorination process, it is at present difficult to say if the cyanide process will eventually prove a serious competitor, as no doubt the process which is the cheapest will maintain itself.

Pyrite (Fe  $S_2$ ) is decomposed by the oxygen of the air and moisture into soluble ferrous sulphate and free monohydrated sulphuric acid, according to the reaction:

$$FeS_2 + H_2O + 7O = FeSO_4 + H_2SO_4$$
.

The ferrous sulphate is decomposed by the action of the air to insoluble basic sulphates. In addition, normal ferric sulphate (Fe<sub>2</sub> 3 SO<sub>4</sub>) is produced, which gradually loses acid

and becomes a soluble basic sulphate, Fe<sub>2</sub> O<sub>3</sub>, 2SO<sub>3</sub>. There are many basic salts of somewhat complex and doubtful composition formed likewise. Thus in an oxidized ore which has contained pyrite are found sulphuric acid, ferrous sulphate, basic ferrous sulphates, ferric sulphate, and basic ferric sulphates, all of which react upon potassium cyanide.

Sulphuric acid reacts upon potassium cyanide with evolution of hydrocyanic acid according to the reaction:

$$2KCy + H_2SO_4 = K_2SO_4 + 2HCy$$
.

Ferrous sulphate reacts upon cyanide with the formation of ferrous cyanide, a yellowish red floculent precipitate:

$$FeSO_4 + 2KCy = FeCy_2 + K_2SO_4$$
.

This ferrous cyanide is attacked by the excess of cyanide in the solution, and ferrocyanide of potassium is formed according to the reaction:

Fe 
$$Cy_2 + 4KCy = K_4FeCy_6$$
.

That is to say, one molecule of ferrous sulphate decomposes or renders inert six molecules of cyanide of potassium. Other things being equal, if 1 per cent. or 20 lbs. of ferrous cyanide existed in the ore, some 51 lbs. of cyanide would be rendered inert for the solution of gold, and, in fact, would be lost. This, at the average price of chemically pure cyanide, would cost over £5 to £6 per ton of ore treated.

The ferrocyanide of potassium, formed according to the last reaction, is reacted upon, if sufficient acid be present, by an additional quantity of ferrous sulphate, with production of Prussian blue according to the reaction:

$$3K_4FeCy_6 + 6FeSO_4 + 3O = Fe_2O_3 + 6K_2SO_4 + Fe_7Cy_{18}$$

This production of Prussian blue gives a blue colour to the surface of the tailings, or to the solution, and indicates at once that the washing and neutralizing operations have not been carried on properly, and that a great loss of cyanide is taking place.

Ferric salts, when present, unmixed with ferrous salts, de-

compose the cyanide solution with formation of hydrocyanic acid and precipitation of ferric hydrate, according to the reaction:

$$Fe_2(SO_4)_3 + KCy = Fe_2Cy_6 + 3K_2SO_4$$
.

With further decomposition:

$$Fe_2Cy_6 + 6H_2O = Fe_2(OH)_6 + 6HCy.$$

This means that, other things being equal, one molecule of ferric sulphate decomposes six molecules of cyanide. If r per cent., or 20 lbs., of ferric sulphate existed in the ore, very nearly the same weight of cyanide, costing  $\pounds_2$  to  $\pounds_3$ , would be destroyed.

If a mixture of ferric and ferrous sulphate, as is probable, exists in partially-oxidized ores, it causes the production, when ferrous sulphate is in excess, of ferrous cyanide, according to the reaction:

$$12KCy + 3FeSO_4 + 4Fe_2(SO_4)_3 = Fe_3(FeCy_6)_2 + 6K_2SO_4$$

when ferric sulphate is in excess, the production of ferric ferric cyanide (Prussian blue), according to the reaction:—

$$18KCy + 3FeSO_4 + 2Fe_2(SO_4)_3 = Fe_4(FeCy_6)_3 + 9K_2SO_4.$$

These reactions show clearly that washing by water and neutralization by a caustic alkali must be employed to arrive at satisfactory and economical results. It is more than probable that many of the failures already recorded are due to the lack of these precautions. In addition to these reactions, there are many with unknown compounds, the composition of which cannot be expressed, even where the greatest precautions are used, and the operations supervised with the greatest ability and knowledge.

Precipitation of the Gold.—Zinc precipitates the dissolved gold, as the cyanide has more affinity for it than for the gold. Theoretical reaction is:

$$2KAuCy_2 + Zn = 2Au + K_2ZnCy_4$$
.

But much more zinc goes into solution than this reaction calls

for. According to the chemical reaction, the consumption should be r oz. of zinc to 6 oz. of gold. The excessive consumption of zinc must be ascribed to other action than the mere replacement of zinc for gold in the double cyanide of gold and potassium.

There is comparatively little exact knowledge of the reactions taking place in the zinc precipitation-boxes. One fact is known positively, and that is, that hydrogen is evolved. This does not occur, however, when zinc alone is exposed to a cyanide solution, but after gold is deposited on the zinc, or when zinc is placed in contact with iron. In other words, a galvanic couple is formed, the water is decomposed, and hydrate of zinc is formed, which is attacked by the cyanide forming a double cyanide of zinc and caustic potash. The probable reactions may be expressed as follows:

$$Zn + 2H_2O = 2H + Zn(HO)_2$$
,  
 $Zn(HO)_2 + 4KCy = ZnK_2Cy_4 + 2KHO$ .

The production of caustic alkali explains the increased alkalinity of the solution after passing the zinc precipitation-boxes. It may be considered advantageous to a certain extent, however, as carbonic acid, which decomposes the solution, is absorbed by the caustic potash, with formation of a carbonate of the alkalies. Ammonia is formed also, as is indicated by the strong odour of the gas about the boxes.

The precipitate contains, besides the precious metals, many of the base metals, which may be dissolved by the solution. The principal of these are copper, arsenic, and antimony. When a weak cyanide solution contains copper, the copper may be precipitated in preference to the gold, whereas, by increasing the quantity of cyanide, the copper can be kept in solution until the precipitation of the gold is complete.

It is also asserted that, in ordinary cases, the accelerating influence on precipitation of excess of potassic cyanide is probably due to generation of nascent hydrogen.

$$4KCy + Zn + 2H_2O = ZnK_2Cy_4 + K_2H_2O + H_2.$$

This nascent hydrogen steps into the place of the gold in the auro potassic cyanide:

$$2AuKCy_2 + H_2 = 2KCy + K_2H_2O + Au_2$$

The hydrocyanic acid thus formed recombining with any free alkali present, there is no loss of such cyanogen as was combined with the gold; from the former of the two equations it would appear that some proportion of the potassic cyanide must be consumed in the zinc boxes. As a matter of fact, there is a consumption in the case of strong cyanide solutions, which, however, in the case of ordinary working (it is claimed \*), when solutions are coming off, 0.2 per cent., or so, Indeed, it would appear as if a reis quite inappreciable. generation of the zinc potassic cyanide took place, the zinc possibly forming a hydrate, and remaining in solution as such, owing to the presence of the free alkali. Given favourable conditions, indeed, the zinc potassic cyanide is itself capable of dissolving gold from ores, and by addition of free alkali to this salt all the cyanogen in it may be determined in the ordinary manner by means of nitrate of silver solution.

Owing to the large amount of zinc which is dissolved in the precipitating-boxes, one would surmise that the working solutions would become in time very highly charged with zinc compounds. Mr. Feldtmann \* remarks on this subject, that, as a matter of experience, it may be stated that they do not, to any great extent, and the probable reason for this is, that the small quantities of alkaline sulphides formed serve to precipitate, at least, a portion of the zinc as insoluble sulphide, a regeneration of potassic cyanide taking place simultaneously.

$$Z_nK_2Cy_4 + K_2S = Z_nS + 4KCy$$
.

The presence, or rather the formation, of alkaline sulphides in the solutions is explained by the action of potassic cyanide on the iron sulphide contained in partially-decomposed pyritous ores.

$$6KCy + FeS = K_4FeCy_6 + K_2S.$$

<sup>\* &</sup>quot;Notes on Gold Extraction by means of Cyanide of Potassium," by W. R. Feldtmann.

Mr. T. S. MacArthur has even found that, in very exceptional cases, sufficient alkaline sulphide may be formed to be of hindrance to the action of cyanide on the gold, and has discovered a remedy for this in the addition of metallic (particularly lead) salts, capable of forming insoluble sulphides.

**Decomposition of the Cyanide.**—The compound of cyanogen and potassium is extremely unstable. Not only is it decomposed by mineral acids and acid salts, but by the action at ordinary temperatures of atmospheric carbonic acid, according to the reaction:

$$2KCy + CO_2 + H_2O = K_2CO_3 + 2HCy.$$

Hydrocyanic acid is given off, a portion of which remains in solution and is available for the extraction of gold, but the greater part is dissipated into the air.

The cyanide is easily oxidized to cyanate:

$$KCy + O = KCyO$$
.

The cyanate is further oxidized to carbonate according to the reaction:

$$2KCyO + 3O = K_2CO_3 + CO_2 + N_2$$
.

The nitrogen given off may cause a still further decomposition, for when a current of nitrogen is passed through a cold dilute solution of cyanide of potassium, hydrocyanic acid is evolved without the nitrogen entering into the reaction. This action, when the presence of a chemical causes a reaction between other chemicals in aqueous solution without entering into the reaction itself, is called hydrolysis, and further reaction must be attributed to this property of caustic alkalies, which are, and must be, always present in a working solution of potassium cyanide. If the solution is boiled with acids or alkalies, hydrolysis of the cyanide occurs rapidly, ammonia and formic acid being formed thus:

$$KCN + 2OH_3 = NH_3 + HCO_2K (+ 9.5).*$$

<sup>\*</sup> According to T. K. Rose, B.Sc., "The Metallurgy of Gold;" London, 1894.

If lime is in solution as cyanide of calcium, the following reaction may explain the formation of calcium carbonate:

$$Ca_{2}(CN) + 3OH_{2} + 2H = CaCO_{3} + 2H_{2}N + CO_{2} + OH_{2}$$

Hydrogen is always evolved when the gold is precipitated; without it the foregoing reaction could not be completed. This reaction accounts, moreover, for the formation of ammonia at the zinc boxes. The carbonic acid evolved attacks the cyanide, as mentioned before, thus causing still greater decomposition of the solvent.

It will be seen, therefore, that the decomposition of the cyanide solution, and loss of the solvent energy of the solution for gold, may be divided under the following heads:—

- 1. Actual decomposition of the solution:
- (a) By acids and acid salts present in the ore.
- (b) By atmospheric carbonic acid.
- (c) By oxidation.
- (d) By reaction owing to hydrolysis.
- 2. The solution of other metals than gold which are not precipitated:—
- (a) By metals or their compounds present in the ore, as oxides or carbonates of lead, which are first attacked by the caustic alkali; by oxides or carbonates of the alkali; by certain compounds of iron insoluble in water.
- (b) By the replacement of gold in solution by zinc in the precipitation.
- (c) By dissolving zinc hydrate, formed by electrolysis, in the precipitation.

It will thus be seen that these losses, under incompetent management, may be frequent, and in many cases may cause unprofitable results.

Testing the Strength of Solutions.—The method employed for estimating the amount of cyanide in a solution is based on the capacity of cyanide of potassium to form a double cyanide with silver—which is added to it in the shape of a titrated solution of nitrate of silver—and on the fact that any

silver solution which is added, beyond the exact quantity which is required, to convert all the potassic cyanide into argentic potassic cyanide, will cause a white precipitate.

$$2KCy + AgNO_3 = AgKCy_2 + KNO_3.*$$

As the combining weights of AgNO<sub>3</sub> and K.Cy. are 170 and 65'13 respectively, it follows that 170 parts by weight of argentic nitrate may be added to 2 by 65'13 = 130'26 parts of potassic cyanide before a permanent precipitate ensues. If, therefore, we add from a burette a solution of argentic nitrate containing 17 grms. in a litre, or 1,000 cubic centimetres, to the solution of potassic cyanide to be tested, until a faint precipitate appears, each c.c. of silver solution added will correspond to  $\frac{13'02}{1000}$  = 0'013 grms. of pure potassic cyanide. From the amount of cyanide solution operated on, the percentage contents can be calculated. It is obvious that the strength of the silver nitrate solution may be so adjusted as to save all calculation.

If, for instance, it is made by dissolving 13.05 grms. of pure silver nitrate in a litre of water, and 10 c.c. of the cyanide solution be taken for a test, then each c.c. of silver solution added will correspond to 0.1 per cent. of pure K.Cy. in the sample tested. In testing very strong solutions—from the dissolving tank, for instance—one-tenth of the quantity of sample may be taken by measuring 10 c.c., diluting with water to 100 c.c., and then drawing off 10 c.c. for test. Of course, in such a case, 1 c.c. of standard silver solution will indicate 1 per cent. of K.Cy. in the original sample of cyanide solution. Addition of a few drops of potassic iodide to the solution to be tested will enhance the accuracy of the test, and will, moreover, annul the danger of our estimating the quantity of cyanide present, consequent on the strong alkalinity of the solution.

In estimating very dilute solutions, such as are employed, for instance, in the Siemens-Halske process, I would recom-

<sup>\*</sup> From Mr. Feldtmann, "Notes on Gold Extraction."

mend the employment of a standard decimal silver solution, as the same would ensure greater accuracy.

By taking 100 c.c. of the ordinary standard solution, diluting the same to 1,000 c.c., 1 c.c. of this decimal solution would be equal to 0'01 per cent. of K.Cy. The strength of the standard nitrate solution should be occasionally controlled by a standard cyanide solution.

In making the analysis, the solutions are filtered through a little quicklime. On addition of the silver nitrate a white curdy precipitate forms, which redissolves, and it is added till the solution shows a white precipitate.

An easier and more accurate method is by titration with a standard solution of iodine in potassium iodide until a blue coloration is apparent, a starch solution having been added to the cyanide solution. The reaction is as follows:—

$$K.Cy + I_2 = KI + ICy$$
.

The solution may be checked on chemically pure cyanide, or, better yet, on sodium hyposulphite, and the cubic centimeters equivalent to I per cent. of potassium cyanide calculated.

If the percentage of zinc in the solution is required, the solution is evaporated to degrees and the residue treated by any of the well-known methods of analysis for zinc.

The cyanide employed is not chemically pure, and the commercial article contains about 80 to 90 per cent. or over. It is customary in some works to dissolve in a small tank several hundredweight of the cyanide, and test the strength of this *storage solution*, and make up from it the stock solutions. The lumps of cyanide are placed on a filter, made of a coarse sieve covered with jute. A pump is used for causing circulation of the solution.

The insoluble impurities, chiefly carbide of iron, contained in the commercial cyanide remain in the tray. The pumping of the solution into this tray should be kept going fast enough to keep the lumps of cyanide covered by solution, as it is found that alternate exposure of the carbide to the air and immersion in solution of cyanide, causes a certain amount of

decomposition of the latter. A water wash may be applied to remove the last trace of cyanide from the carbide before throwing the latter away. On no account should this carbide be put on the top of the sand in the filter vats, as is sometimes done.\* Having determined the strength and quantity of the stock solution, and the strength of the solution in the dissolving tank, the following is a simple formula for arriving at the quantity of the latter requisite to bring the former up to the desired strength:—

```
A being desired strength of stock solution in per cent.;

B being present strength of stock solution in per cent.;

C the strength of dissolving tank solution in per cent.;

D the quantity in tons, lbs., gals., litres, &c., &c., of stock;

A - B
C - A × D = quantity of dissolving tank solution to be added (in tons, lbs., gals., litres, &c., &c.)
```

For example, supposing the stock solution to consist of 100,000 galls. of 0.4 per cent. strength, and it be desired to bring this up to 0.6 per cent. by adding some 10 per cent. solution, then—

$$\frac{0.6 - 0.4}{10 - 0.6}$$
 x 100,000 = 2127.65 gals, of the dissolving tank solution,

The following table (pp. 86, 87), which was kindly prepared for me by Mr. Blomfield, chemist of the works, will exemplify the strengths of the different solutions at the various stages while leaching from tailing-vat (this particular lot, which was treated at the Rand Central Ore Reduction works, being somewhat slimey, took a longer time to treat, i.e., about 60 hours):—

See Feldtmann on "Gold Extraction."

Time of Leaching.	Original Strength of Solution.	Strength of Solution after Contact-	Remarks.
I hour 3 hours 5 ", 7 "	st Weak Solution 0·16 per cent.	0.05 per cent. 0.05 ,, 0.075 ,,	There was considerable moisture in the tailings previous to running on the 1st weak solution.
After 1 hour. ,, 3 hours. ,, 5 ,, ,, 7 ,, ,, 9 ,, ,, 11 ,, ,, 13 ,,	Strong Solution. 0°35 per cent.	O·I per cent. O·2 ,, O·25 ,, O·24 ,, O·26 ,, O·25 ,, O·26 ,, O·25 ,, O·3 ,, O·24 ,,	Draining dry.

Time of Leaching.	Original Strength of Solution.	Strength of Solution after Contact.	Remarks.			
	0.16 per cent.					
After I hour.	•	0.26 per cent.				
", 3 hours.		0.3 "				
,, 5 ,,		0.33 "				
2nd Weak Wash.						
,	0.16 per cent.					
I hour.	-	o·33 per cent.				
3 hours.		0.3 "				
5 », 7 »		0.53 "				
7 "		0.55 "				
	3rd Weak Wash	•				
	0.16 per cent.					
I hour.	-	'0.2 per cent.				
3 hours.		0.17 ,,				
5 ,,	,	0.12 "				
	4th Weak Wash.					
	0.16 per cent.	l				
I hour.	•	0.15 per cent.	,			
3 hours.		0.14 ,				
5 ,,		0.12 "				
Final Water Wash.						
I hour.	1	0.15 per cent.				
3 hours.		۰٬۲ ,,				
5 ", 7 ",		0.13 "				
7 ,,		o-05 "				

Determination of Gold in Cyanide Solutions.— Buchanan's method consists in precipitating a known quantity of solution with excess of argentine nitrate, decomposing the precipitate formed by means of a reducing agent, filtering, drying, and cupelling direct. In detail he found the best method of procedure to be as follows:—195 c.c. of the cyanide solution are transferred to a flask of about 500 c.c. capacity, and mixed with a few drops of potassic chromate. Argentic nitrate solution of any convenient strength, say, 5 per cent., is then added until the characteristic reddish tinge of the chromate shows that the reaction is complete. Then take 10 to 20 grms. of zinc dust or shavings, mix them thoroughly with the precipitate and solution in the flask, and add 2 or 3 c.c. of 10 per cent. sulphuric acid. Allow to stand for 10 minutes, add excess of sulphuric acid to dissolve the remainder of the zinc, filter, wash once, dry, and incinerate on a roasting dish in the muffle and cupel with a little lead. This method does away with the trouble of fluxing, and has the advantage of allowing of a great number of samples being done together. The results on the whole are slightly lower than those obtained by the ordinary precipitation method.

Crosse's method consists in taking  $\frac{1}{4}$  or 1 litre of cyanide solution, containing gold, and adding excess of nitrate of silver. A precipitate of cyanide of silver is formed, and also argento-auro-cyanide, which is insoluble. This precipitate quickly settles, and is filtered on to a large filter. It is then put in a crucible covered with flux and, say, 500 grs. of litharge. In 10 minutes the fusion is complete, the button of lead obtained is cupelled, and the gold parted from the silver. This method allows of a larger quantity of liquid to be operated upon, is easily carried out, and does not require much watching. In the case of strong cyanide solutions most of the potassic cyanide is decomposed by addition of acid previous to adding the silver nitrate, to avoid the otherwise heavy consumption of the latter.

Extraction Tests.—Mr. Feldtmann gives the following

instructions as to the best tests for determining if an ore is amenable to cyanide treatment.

It is assumed that the total sample is crushed fine enough to pass a 30-mesh sieve.

- 1. Assay a portion of the sample.
- 2. Determine the amount of cyanide it will consume by shaking test. For example: 200 grms. of ore are placed in a glass stoppered bottle with a 100 c.c. of solution of cyanide of 0.5 per cent. strength, and shaken for twenty minutes or so.

A portion of the solution is then filtered off and tested. Supposing it to be reduced in strength to 0.4 per cent., showing a consumption of 0.1 per cent. on the solution, or half as much, i.e., 0.05 per cent. on the ore (or 1 lb. per ton), we might safely conclude that the ore will not require any preliminary treatment before leaching with cyanide. The largest consumption of cyanide takes place almost immediately after the solution comes in contact with the ore, and after twenty minutes' shaking it is generally safe to assume that there will not be much further consumption.

3. If it is found that the consumption of cyanide is excessive, a third portion of the ore is tested for "cyanicide," by which is meant free acid, soluble and basic iron salts, and indeed any cyanide destroying substance which may be coun-A solution of caustic soda of known teracted by alkali. strength is run, little by little, from a burette into a weighed quantity of the ore mixed with water, the mixture being well stirred after each addition of alkali, until a drop, taken out on a glass rod, will just turn red litmus slightly blue. A convenient quantity of ore to operate on is 200 grms., and, using an alkali solution of 10 grms. commercial caustic soda to the litre of water, each cubic centimetre will correspond to 10th lb. of the same quality caustic soda required to the ton (of 2,000) of ore. If the consumption of soda be more than 3 lbs. per ton, it will generally be found advisable to water wash the ore before giving alkaline treatment. to determine the amount of alkali which may be saved through a preliminary water wash, by first estimating total "cyanicide,"

then taking another sample, water washing first, and estimating remaining "cyanicide."

It is usual, when reporting the amount of "cyanicide" in an ore, to do so in terms of pounds and fractions of caustic soda required to neutralize a ton of the ore.

Should the consumption of cyanide in No. 2 test have proved larger than the amount of iron salts and acid present would account for, there is probably copper in the ore. The cyanide solution from test No. 2 may be conveniently examined for copper by evaporating with nitric acid, taking up with a little more nitric acid, diluting and precipitating with ammonia, when copper will be indicated by the characteristic blue colouration of the liquid.

4. Several samples of ore are weighed out for extraction tests. A suitable vessel for testing extraction is a lamp glass, fitted with an india-rubber stopper, with a glass tube through it, which may be closed by means of a small piece of rubber tubing and burette clip. A filter is formed over the rubber stopper by means of a piece of sponge, some filter-paper, or some asbestos fibre.

Into such vessels the samples are placed—say, 200 grms. each—and they must then receive whatever preliminary treatment test No. 3 has shown to be needful in the way of water and alkali washes, 100 c.c. of a 0.5 per cent. solution of cyanide is then poured on. The various samples may be allowed to stand for different periods, say, one, two, and three days respectively, or more, if thought fit; the cyanide solutions being then drawn off and tested, and the ore, after water washing to remove all dissolved gold, being assayed again.

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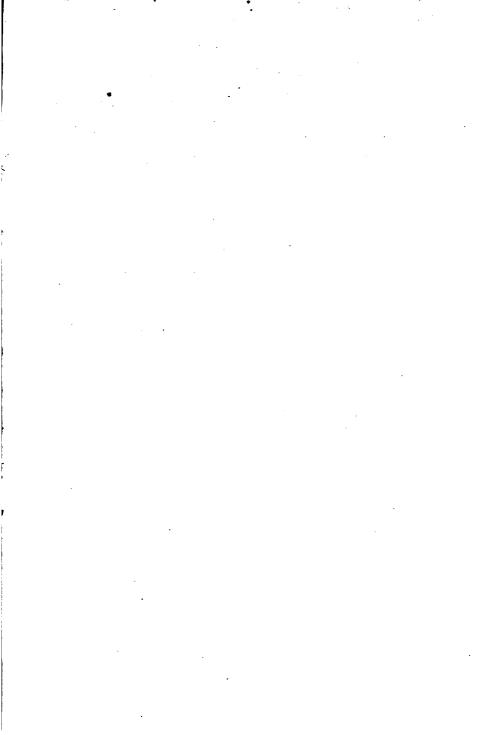
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